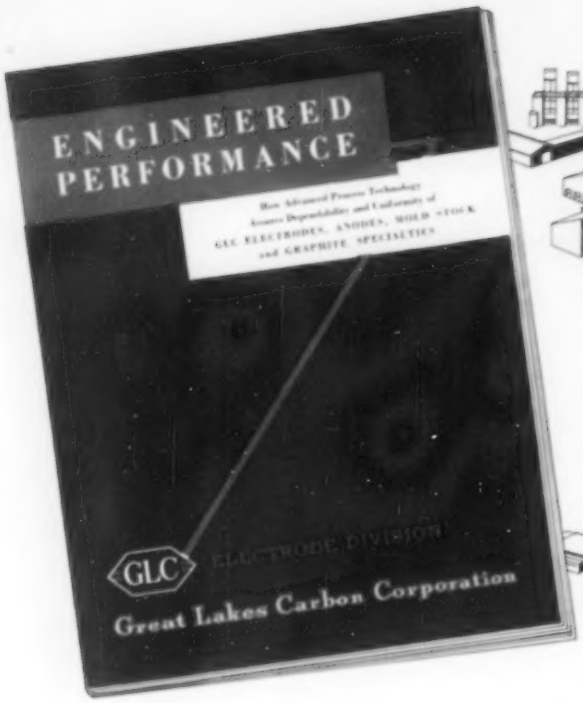
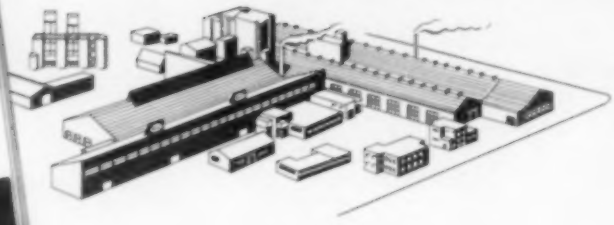
JOURNAL OF

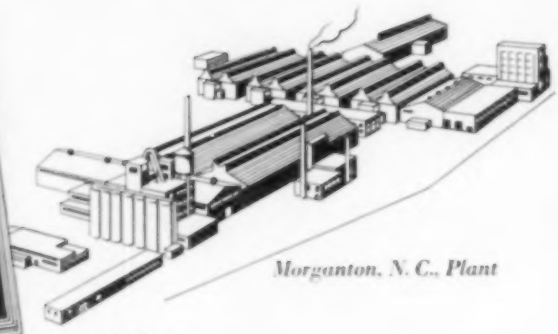
METALS

July 1955



Niagara Falls, N.Y., Plant





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The AIME Publishes:
Journal of Metals
Mining Engineering
Journal of Petroleum Technology

VOL. 7, NO. 7 JULY, 1955

COVER

The importance of coal to furnish power for the production of aluminum is stressed in the article beginning on p. 796. Harry Brocke illustrates this theme on the cover.

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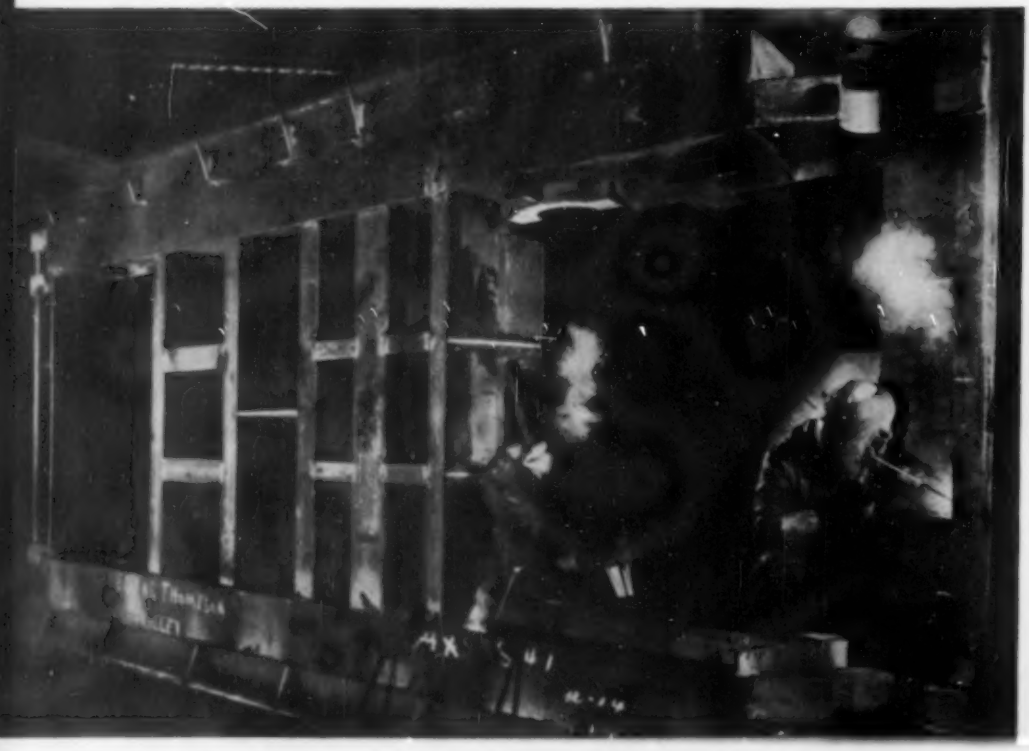

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Research Engineer, 35 to 40, to take charge of research laboratory, mostly chemical and metallurgical operations and some work in sanitation, that is, sewage disposal, refuse disposal for pulp and paper industry. Metallurgical calcining and chemical operations. Must have had experience in this field and have ability in management. Salary open. Location, northern New Jersey. W1514.

Physical Metallurgist, Ph.D. met. eng., up to 50. Must have had at least two years experience as physical metallurgist and have knowledge of ferrous and nonferrous metals including titanium. Will set up program to include metallography and mechanical properties of titanium alloys. Company manufactures valves. Salary up to $10,000. Location, Chicago. C3160.

Plant Superintendent, 35 to 45, graduate I.E. or M.E. or metallurgical engineer, with at least 10 years experience with general jobbing shop of the boiler shop type, where alloy and code construction was practiced. Experience with alloys such as, stainless steel, Monel, nickel, copper, Hastelloy, etc., very desirable. Familiarity with the application of power tooling for shaping and forming operations, slab work, machine shop practice, welding by metallic arc, submerged gas (heliarc and sigma) and unionmelt is required. Salary, $9000 a year, plus other benefits. Location, California. W1438-S.

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Metallurgical Engineer with three to seven years experience in hydrometallurgy, for an expanding lithium chemical processing plant. Salary open. Location, South. W1452.

Process Research Metallurgist, recent graduate metallurgist, or mechanical engineer with metallurgical interest, for brass and copper alloy mill. Opportunity to gain broad knowledge of entire road, strip, tube and wire mill operations while working. Location, New England. W1483.

Metallurgist, up to 40, for manufacturer of forgings and rolling mill. Must have had at least two years experience in administrative and supervisory work on ferrous metals alloys in production functions on job shop forgings. Will assist chief metallurgist on test work laboratory, customer contact and allied functions on ferrous forgings. Salary, $6600 to $7800 a year. Employer will pay placement fee. Location, western Chicago suburb. C3079.

Corrosion Engineer, metallurgical or chemical engineering graduate, with cathodic protection experience on pipe lines. Salary, $9600 a year, plus $300 a month living expenses. Duration six to eight months. Location, France. F1487.

Chief Design Engineer, mechanical graduate, 32 to 42, with at least 10 years equipment experience covering sheet metal products and controls. Salary, $6000 to $8000 a year. Location, Newark, N. J. area. W-1682(a).

Sales Engineer, with mechanical or metallurgical engineering training and sales experience in metal products fields to sell powder metallurgy products. Salary, $4800 a year plus commission and expenses. W-1637.

Metallurgists for production and research problems connected with stainless steel operations. B.S. degree in engineering required. Salary, open. Location, East. W-1565.

Junior Engineer, mechanical, electrical, or metallurgical graduate for experimental work. Salary, about $4420 a year. Location, northern New Jersey. W-1517(d).

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Metallurgist, recent graduate, no previous industrial experience required to work in materials control laboratory of automotive company on product control and material specifications. Salary, $4800 a year. Location, Michigan. W-1220.

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Books for Engineers

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Coordination, Control, and Financing of Industrial Research, edited by Albert H. Rubenstein. *King's Crown Press,* $8.50, 429 pp., 1955—The report of the Fifth Conference on Industrial Research sponsored by the dept. of industrial engineering at Columbia University comprises the major portion of this book. The fourth conference, held in 1953, is also given space but to a lesser degree.

Turboblowers, by A. J. Stepanoff. *John Wiley & Sons, Inc.,* $8.00, 377 pp., 1955—This book deals with the hydrodynamic and thermodynamic aspects of the turboblower design. It outlines new methods of attack on turbomachine problems, and discusses the art of building turbocompressors in the U. S. and abroad. European designs are given prominent play in the book. New methods and techniques are explored. A theoretical treatment of a single pattern of flow is presented, and extended to treatment of similar design procedure for centrifugal and axial flow impellers.

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X-Ray Diffraction by Polycrystalline Materials, edited by H. S. Peiser, H. P. Rooksby, and A. J. C. Wilson. *The Institute of Physics,* approx. $9.00, 725 pp., 1955—The book is an attempt to answer the need for a comprehensive survey of the special and important practical methods of investigating crystals in powdered or polycrystalline condition. It has been written by some 30 members of X-ray div. of the Institute of Physics, London. Early chapters deal with tactics, and the latter with strategy. The book is divided into three parts; experimental techniques, interpretation of data, and practical uses of the techniques in different fields.

Forming of Austenitic Chromium-Nickel Stainless Steels. *International Nickel Co., Inc.,* $5.00, 394 pp., 1955—The revised and expanded edition of a book that was first published in 1947, the current volume shows the minute details of how to form the chromium-nickel stainless steels. Descriptions of newly developed processes, especially those using only one solid die, and their application to the forming of stainless steels are included, along with detailed discussions and illustrations of older and more conventional methods. The book explains the effect of various compositions, tempers and finishes on the forming characteristics.

Manganese, by A. H. Sully. *Academic Press, Inc.,* $6.50, 305 pp., 1955—This is the third book in the series Metallurgy of the Rarer Metals. The book, the author believes, contains a description of all the more important contributions which have been made to the study of manganese and its alloys. Conservation and recovery of manganese is given special attention in the early chapters of the book.

Plastics Tooling, by Malcolm W. Riley. *Reinhold Publishing Corp.,* $2.50, 123 pp., 1955—Plastics tooling is still in its infancy. Thus, instead of attempting to present a definitive book on the subject, the author has compiled a status report. The book intends to bring the reader up to date on the resins and methods used, and the major applications of plastics tooling.

The Fabrication Properties of 16-Alfenol—A Non-Strategic Aluminum-Iron Alloy. *U.S. Naval Ordnance Laboratory,* 75¢, 27 pp., 1955—The report describes methods for fabricating 16-Alfenol to overcome its inherent hard and brittle characteristics. The preparation of the alloys, fabrication of ingots into strip and tape, and the magnetic properties obtained are discussed. Please order from OTS, U. S. Dept. of Commerce, PB111552, Washington 25, D. C.

High-Resolution Autoradiography for Study of Grain Boundaries in Metals. *Engineering Research Institute, University of Michigan, for U.S. Army Ordnance Corps.* $2.25, 82 pp., 1955—Research was directed first toward development and refinement of the autoradiography technique and, second to the study of several distinctly different metallurgical systems by autoradiography to evaluate its capabilities and limitations. Results obtained amply demonstrate the value of autoradiography as a metallurgical research tool. Discussed are radioactivity and its detection, the unique value of autoradiography as a means of detection, description of the techniques of making autoradiographs of metallic samples, and descriptions of experimental methods used in making radioactive specimens for each of the systems studied and autoradiographic results obtained. Metallurgical phenomena studies include diffusion, gases in metals, effect of heat treatment on mode of distribution of a minute phase, and the modifying effects of trace quantities of an element on the formation of a second constituent. Please order from OTS, U. S. Dept. of Commerce, PB111558, Washington 25, D. C.

Consulting Services, 15th edition. *Assn. of Consulting Chemists and Chemical Engineers, Inc.,* $1.00, 144 pp., 1955—Section I of this book contains more than 200 items of activity, with the members specializing or qualifying in the given field indicated by key numbers. Section II describes each member's qualifications and activities. Section III contains an alphabetical listing of members, and names of organizations with which they are connected by geographical location. Letterhead requests stating source of reference.

Guide for Labour Inspectors. *International Labour Office,* $1.00, 107 pp., 1955—In drawing up the guide the International Labour Office took into account the International Labour Conference at the Labour Inspection Convention, 1947. While practical inspection standards have been evolved on the international plane it has been recognized that details of the organization and operating practices of inspection services vary from country to country.

Depreciation, by E. L. Grant and P. T. Norton, Jr. *Ronald Press Co.,* $7.50, 504 pp., 1955—This revised comprehensive manual for engineers, accountants, and industrial managers deals with depreciation from the point of view of accounting and income taxes. This edition includes a new chapter on the Internal Revenue Act of 1954 as it affects the rules governing depreciation tax practice.

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Industrial Notes

• *Kaiser Aluminum & Chemical Corp. is installing $750,000 worth of aluminum casting facilities at its Mead, Wash., reduction plant. Construction is underway and completion is scheduled for late this year. Casting facilities, using the direct chill method, will supply rolling ingot for Kaiser Aluminum's nearby Trentwood sheet and plate rolling mill and other potential customers.*

• The Atomic Energy Commission approved a regulation establishing procedures and criteria for granting permits for access to confidential and secret restricted data relating to civilian uses of atomic energy under a program announced Apr. 20, 1955. Data of military significance is not within the scope of this program. Under the *access permit* program confidential restricted data may be made available to any person who can show a potential use or application of the information in his business, profession, or trade. However, secret data may only be made available to persons showing evidence of need, limited to a definite period of time, for specific information having an immediate or significant effect on their business, profession or trade.

• A 32-page dictionary of steel terms of particular value to men who buy or use cold finished steel bars has been published by LaSalle Steel Co., Chicago 80, Ill. Called *Simplified Steel Terms and Engineering Data*, **the book contains more than 180 relatively detailed definitions frequently used in the purchase, manufacture, treating, machining, and finishing of steel.**

• The 13th Annual Pittsburgh Diffraction Conference will be held November 3 and 4 at the Mellon Institute, Pittsburgh. Sessions will deal with instrumentation and methods, neutron diffraction, and metals, with provisions for papers on general diffraction subjects. Emphasis this year will be on imperfections and phase transformations. Titles of contributed papers should be sent to the program chairman, W. L. Kehl, Gulf Research & Development Co., P.O. Drawer 2038, Pittsburgh 30, before September 1. Abstracts should be submitted by September 20.

• *General Services Administration granted a $1,183,495 contract to National Research Corp. to finance construction and operation of a demonstration pilot plant for production of titanium metal by a new non-Kroll process developed by the company. The plant will be designed for 1000 lb of metal per day and will be housed in a building to be erected near the company's equipment div. in Newton, Mass. The process has already been operated on a small pilot plant scale in Cambridge for two years under a joint program with Monsanto Chemical Co., producing a metal of high purity in crystalline form.*

• American Silver Co. predicts that it will soon be rolling titanium at a thickness of 0.0005 in. The company has already produced commercial quantities of rolled titanium down to 0.0008 in. American Silver produces titanium foil in the commercially pure alloys, Ti-75A and RC-55.

• Total estimated employment in the iron and steel industry rose 13,300 in April over March figures, with some 643,200 persons at work. Wage earners averaged 39.9 hr per wk during April, with a total estimated payroll of $282,025,000 for wage earners and salaried personnel.

• *Norelco X-ray Diffraction Schools, sponsored by the research & control instruments div., North American Philips Co., Inc., are scheduled at the Sir Francis Drake Hotel, San Francisco, September 26 to 30, and at the Hotel Knickerbocker, Chicago, October 10 to 14. No registration fee is charged for the schools.*

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AUTOMATION is getting to be almost as familiar a word as ain't. The latest thing is a driverless tractor train that can be summoned almost anywhere in a plant by simply pushing a button. The latest in material handling magic was shown at the National Materials Handling Exposition in Chicago, when a fully loaded tractor train made its way through a plant to the shipping room. The train follows a route set by a guide wire strung from the ceiling. It's called the Guide-O-Matic, and is the latest development of the Barrett-Cravens Co. The setup is completely automatic with neither operator nor physical connection required for it to work. The load moves steadily toward its terminus at any chosen speed, completely under automatic control all the way.

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RECENTLY, the Babcock & Wilcox Co. opened the doors of its research center at Alliance, Ohio to the press. A tour through the installation convinced the journalists that research with a capital "R" is the order of the day for the people who work there. The staff constantly probes for the answers to a variety of problems—and the answers they come up with eventually result in new products, plants, and production lines.

One of the most significant projects at the research center involves both high pressure water cooled reactor and liquid metal cooled reactor systems. Water cooled system studies involve heat transfer, corrosion of various stainless *structural* alloys, and corrosion of fuel element cladding materials. The possible use of carbon steel and other economically available construction materials is under investigation for commercial power reactor plants. One of the processes developed at the research center is for the analyses of carbon and oxygen in sodium and sodium-potassium alloys. Other projects include proof tests for proposed heat exchangers, pressure vessel stress studies, closure tests, material tests, and thermal shock tests.

A coordinated program at the center has been worked out for development of new boiler plants for high temperature and pressure operation through research into melting, casting, forging, and machining new metal alloys. Experimental alloys are being prepared and tested for high and low temperature corrosion and creep-rupture. One program extending over more than two years involved more than 245 different alloys, with thousands of hours expended for testing. While the end result was only five usable alloys, it contributed to the development of new and better equipment.

The foundry at the center is equipped for melting, casting, forging, and heat treating alloys. Four induction furnaces, powered by a 50 kw, 3000 cycle motor generator produce melts from 10 to 140 lb. Ingots are forged on a 2500-lb steam hammer and a smaller air-operated unit. Within the foundry research area are facilities for preparing and testing sand molds. When completed standards for spectrographic tests of alloys are expected by B&W to be the most comprehensive in existence.

The metallurgical section of the laboratory is gathering material on the mechanical properties of metals at low and high temperatures, in addition to doing research on strength, ductility, toughness, and shock and impact resistance. The section has facilities for heat treating, micro-polishing, micro-examining, and for determining the expansion behavior of metal alloys. Alloys can be heat treated at temperatures up to 2350°F for short periods and at more usual service temperatures in the neighborhood of 1850°F for longer periods. Microstructures of metals can be investigated at magnifications up to 2500X.

Other sections are at work on engineering physics, welding, refractories, chemical problems, oil and gas burning, steam generation, and cyclone furnace utilization.

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STEEL is no longer a "prince or pauper" business, according to Arthur B. Homer, president of Bethlehem Steel Co. Speaking at the recent annual meeting of the American Iron & Steel Institute, Mr. Homer predicted an average increase in capacity of the steel industry of about 4 million tons a year over the next 15 years. Thus, by 1970 capacity would be approximately 50 pct greater than it is today.

Behind Mr. Homer's thinking is a predicted population rise in the U. S. to 200 million people by 1970, with an ever increasing rise in the per capita steel consumption. Along with his prognosis, Mr. Homer expects that the current "modern conservative" policies of the industry will continue on the part of the individual companies, the public, and Government. The steel executive makes no bones about 1954. It was a year "of unmistakable recession as reflected by some measuring rods," he feels.

"What happened was a spiraling down which was arrested . . . The economy leveled off at a relatively high plateau and the nation was back on the road to recovery which we are still traveling. I believe . . . this reflects the operation of a number of plus factors in our economy which give grounds for confidence that we are on the threshold of a period of stabilized growth.

"Such stabilizers as unemployment compensation, pensions, social security, and veterans' payments performed their role in maintaining spending lev-

els." Those factors, plus a reduction of personal taxes, meant that "disposable income after taxes actually increased during a time of technical depression."

Mr. Homer made note of the progress in respect to labor matters. "Those of us in the industry know that the campaign for more income security for hourly-paid employees has obscured the very real gains which have been made on that front. The steel industry's expenditures since 1946 of over $6 billion for modernization and expansion have greatly broadened our product ranges and have enabled us to offer a far greater degree of employment stability than ever before . . . The billions spent to diversify our products and therefore our consuming fields will continue to be an immense factor in stabilizing employment."

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CANADIAN aluminum reaching fabricators in the U. S. has shown a steady increase since 1949. This year some 225,000 tons are expected to flow into this nation. Aluminium Ltd. has been energetically wooing the independent fabricators, despite the 1.5¢ per lb duty it must absorb. Aluminium Ltd. sold to only seven U. S. fabricators in 1946, but today is serving more than 100 firms. What's more, shipments to the U. S. are expected to jump to a total of 290,000 tons in 1956, according to Nathanael V. Davis, president of Aluminium Ltd.

Mr. Davis, in a recent appearance before the House Small Business Subcommittee, stated: ". . . we have . . . solicited the business of the rapidly growing independent (nonintegrated) fabricators. We have done so in the belief that the independent fabricators would look upon us as natural supplier, not competing with them in the U. S. fabricating business—and that we on our part could look to them as steady buyers in both good times and bad times—in comparison with those who produce their own ingot requirements."

Mr. Davis noted that Aluminium Ltd. cannot fill all the orders received by his company, but hopes that the future will bring even greater increases as the projected expansion program brings in more production.

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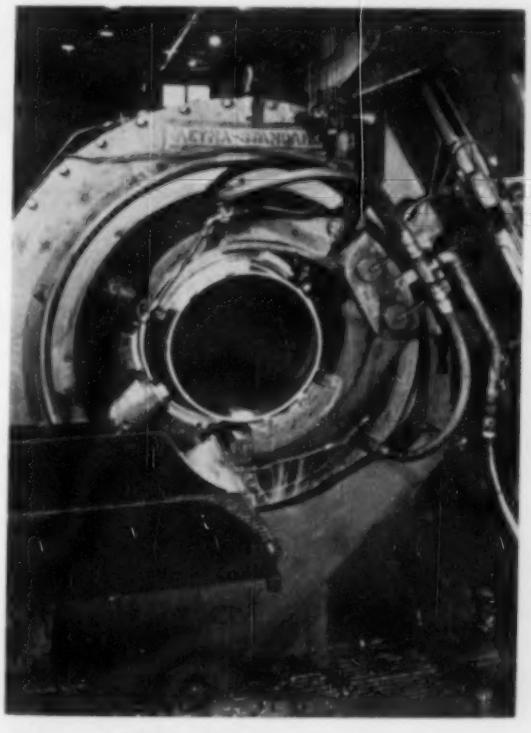
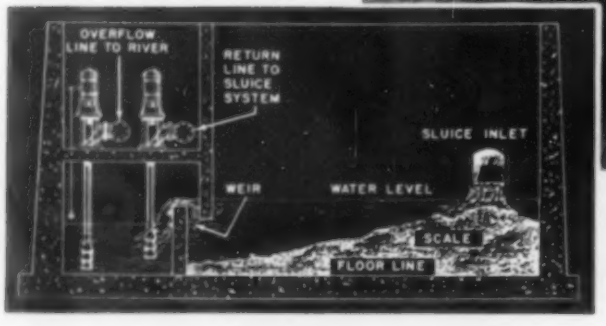
NO one has been blowing trumpets or beating drums but in the last 10 years the Netherlands has developed its metals industries to a remarkable degree. Until recently, the country has been primarily an agrarian nation, but in the last decade the metals industry has expanded to a greater extent than any other segment of the economy. It necessitated a reorientation of the working force—for years undermined by self-imposed passive resistance against the German occupation authorities. In addition to the general lassitude, the Netherlands had a scarcity of foreign currency, making it difficult to replace machines taken by the occupation forces. The industry had regained its prewar standard by about 1948 or 1949 and in the five years since 1949 the Dutch metals industries increased its sales by about 130 pct. Aggregate Dutch industrial sales increased 95 pct for the same period.

By 1950 Dutch metals industries employed 423,000 persons, just about twice the number in 1930, although population increased by only 28 pct between those years. By the end of the fourth quarter of 1954 the metals industries were pushing hard against the food industry—for centuries the leader in Dutch economic life. Aggregate turnover of the metals industries at that time reached an annual rate of almost 7 billion guilders, compared with the country's total industrial turnover of 24 billion guilders annually. Some 30 pct of Dutch exports are metal products. Holland is still in the infant stage of steel production, however. Production is barely 1 million metric tons of crude steel per year. Yet, even that figure is some 18 times the production in the last year before World War II.



A rotating cutoff machine at Lorain works of U. S. Steel Corp.'s National Tube div. prepares a length of seamless pipe for cold extrusion. The machine is held by clamps while six cutting tools in the rotating head trim the pipe end. Extensive research and development was conducted before production began.

Pumping Pit with Ingersoll-Rand vertical turbine pumps, heart of mill-scale recovery system at Allenport, Pa. plant of Pittsburgh Steel Corp. Two 6000 gpm units, in foreground, recirculate settled water. Smaller 3000 gpm pumps, at top, discharge overflow into river.



In Rolling Slabs at the hot strip mill, scale is removed from hot steel by water jets. Water carries the scale into a collecting tank. From the tank, a clam-shell bucket removes it to sintering plant. Steel content is used in blast furnaces, and stream pollution is reduced by the mill-scale recovery.

In mill-scale recovery

I-R Pumps of Stainless give 30 Times the Life

CORROSIVE ACTION of a notoriously unclean river . . . polluted with acid mine drainage . . . is defeated by these Ingersoll-Rand pumps.

Erosion is minimized, too. Erosion by abrasives in recirculated water of a millscale sluicing system. And erosion by hard, tough particles in water agitated by a clamshell used for scale removal.

That's why pumps of standard materials needed extensive repairs after only two months' use.

In contrast, chromium-nickel stainless steel units operated five years without replacements or loss in efficiency and performance.

Ingersoll-Rand pump specialists, metallurgists and engineers use these money-saving nickel alloys for impellers, casings, shafting, bolting and even for small interior parts of pumps in mill-scale service.

You, too, may set new standards of low-cost performance. Redesign to increase strength, yet reduce size and weight of your products or equipment, by using nickel alloys.

When you have a metal problem, let us help you. We'll give you suggestions based on wide practical experience. Take the first step now . . . write for "List A" of our available publications. With it comes a form for outlining your difficulties. Write today.

INCO TRADE MARK

THE INTERNATIONAL NICKEL COMPANY, INC. 67 Wall Street New York 5, N. Y.

Heppenstall Co. announced the first press forging of uranium on a pilot plant commercial scale. A 1000 ton press is used to forge the metal by means of V dies and swage dies. Uranium is heated to forging temperature in a salt bath and quenched in water after forging to prevent oxidation.

Westinghouse Electric Laboratories have developed a method for obtaining high purity titanium for research work. Called cage zone refining, a bar of impure titanium is melted progressively from end to end. Iron, a common impurity in titanium, concentrates in the liquid titanium and follows the molten zone to the end of the bar. More impurities are removed each time the process is repeated.

Aluminum Co. of America announced a new alloy offering excellent properties at 500 to 600°F is available in experimental quantities. The temperature range gives promise of value in the aircraft and automotive industries.

Some 21 U.S. firms will take part in the United Nation's August conference on peaceful uses of atomic energy. They will set up their products in a private exhibition hall in Geneva. The number of U.S. firms contrasts sharply with the government backed British commercial exhibit that includes some 100 United Kingdom manufacturers of nuclear equipment. Official scientific exhibits of the participating nations will be housed in Palais des Nations, four miles from the commercial exhibit hall.

Molybdenum Corp. of America and Kennecott Copper Corp. will jointly develop a columbium-tantalum ore deposit in Canada at Oka, about 43 miles southwest of Montreal.

Plans have been completed and a charter issued for construction of Canada's first seamless steel pipe mill employing the automatic process, according to Mannesmann A.G., Duesseldorf, Germany. The plant is scheduled for construction at Sault Ste. Marie, Ontario. Steel requirements will be filled by the neighboring Algoma Steel Corp.

The Government diverted 200 million lb of aluminum and 4 million lb of nickel from shipment to the national stockpile in the July-August-September quarter to ease continuing shortages. Diversion of aluminum is really a deferment and the Government will obtain the metal at a future date.

It has been reported that some copper consumers have been forced to pay substantial premiums for relatively small tonnages and also to resort to costly conversion deals in copper scrap. Some copper products manufacturers are said to have paid 44¢ per lb for small lots.

Drift of Things

ONE of the fundamental reasons for the existence of the Engineering Manpower Commission is the necessity for establishing the importance of the engineering profession to the national economy. It's a big job and one that must be worked at constantly. The group, a part of Engineers Joint Council, has been in a large measure responsible for the growing awareness on the part of American industry and business of its important stake on the nation's technological manpower potential.

Engineering Manpower Commission was established in 1950 in response to a request from the Manpower Office of the National Security Resources Board to prepare a program "for the most effective utilization of engineers in the national effort and make recommendations as to how a program should be best administered." When the job was finished EJC decided to continue EMC because the shortage of engineers and scientists and the problems made acute by the international situation cried for some kind of action. Three basic aims were set up:

1. To aid in maintaining the supply of trained engineers.
2. To promote the most effective utilization of engineers in the maintenance of the national health, safety, and interest.
3. To aid in establishing the importance of the engineering profession to the national economy.

Who does the work of the Engineering Manpower Commission? Delegates of the professional societies representing the societies constituent to Engineers Joint Council. Money comes from private contributions from those who are interested in the contribution of engineers to the nation. Last year, EMC coordinated its activities with those of the Scientific Manpower Commission, thus assuring that the engineering and scientific professions would be served without wasteful duplication. Funds are now solicited jointly, and with the EMC office in New York and the SMC bureau in Washington, D. C., a degree of flexibility has been achieved that has been heretofore impossible.

One of the major problems faced by EMC has been the steadily declining engineering enrollment throughout the country. In 1951 the commission started its "Engineers Wanted" campaign and by the fall of 1954 it managed to reverse the trend. While the increase in enrollment hardly meets projected demand, the statistics, still problematical, indicate a slight percentage increase has taken place.

But the battle isn't even half won. Evidence that high school students are passing up science and mathematics courses continues to pour in. EMC in 1954 tackled the problem and appointed a group to evaluate the roll of the high school teacher of science. The program of action outlined by the committee is being carried out at this moment, and we can expect to see it bear fruit in the near future.

AIME members who are part of EMC are H. K. Work, L. H. Hart, Julian Feiss, and E. H. Robie.

One of the many functions of EMC is the dissemination of information to individuals. Countless articles, speeches, editorials, news stories, and other forms of communication have been based on information furnished by the commission. Often enough, source credit is omitted, but the message is there—of that there is no doubt.

Probably the most taxing of commission tasks is the promotion of the most effective utilization of engineers. America's technological manpower resources are limited. It represents a new concept—one that is only now gaining some degree of understanding. It involves more than just the military and industrial present. It encompasses the very industrial future of the country.

One of the most valuable instruments in supplying the latest information on demand and distribution of engineering manpower and the basic factors in the engineering manpower situation has been the Newsletters distributed by EMC. The commission has been steadily collecting data on the utilization of engineers in the armed forces and brings the information to the proper people in the Defense Dept.

The group of engineers being separated from the military service possibly represents one of the most significant recruiting sources for new talent. EMC has been unsuccessful in gaining formal cooperation from the Defense Dept. in setting up a system to ease industrial recruiting of these men. It has been able to work with local area commanders through the permissive local option policy of the Defense Dept.

The Engineering and Scientific Manpower Commissions have also pioneered in setting up committees of professional personnel to advise local selective service boards in cases of occupational deferment. The committees were established in six states by the Director of Selective Service on recommendations made by the commissions.

EMC has been doing an important job on the overall picture, as attested by this letter received at AIME headquarters. "I am writing to commend the work of the Engineering Manpower Commission, and the AIME for supporting their endeavors.

"Last summer while working on my Phd. in physical metallurgy at Columbia University, I was drafted into the U. S. Army. Since then I have twice been in considerable difficulty with the army in regard to proper stationing and utilization of my professional background. It has been solely through the efforts of Mr. W. T. Cavanaugh, of the E.M.C. that I am presently properly stationed.

"During these efforts I had the additional opportunity to observe the commission at work in Washington, D. C. before Congress and elsewhere in their efforts to improve the relationships between engineers and the military services.

"I highly commend the work being done by the Commission and as a member of AIME am proud to know that the Institute is supporting the Commission's efforts.

Yours truly,
Walter E. Bron."

A. S. Cohan

Thirty-One Year Old Furnace Bottom Yields Copper Bonanza

by Harold Foard

IN JUNE 1952 bottom soundings were taken of Miami plant's No. 4 furnace which had been in use since April 1922. These soundings showed an estimated inventory of 1,045,096 lb of copper below the floor level. By the end of May 1953 that inventory had increased to 1,206,668 lb. Despite careful attention to fettling, soundings of December 1953 showed a pool of molten matte, 8 ft in depth, to exist at a point 20 ft in front of the bridge wall, as shown in Fig. 1.

During 31 years, the furnace had been shut down many times and upon each occasion the bottom cooled and shrinkage cracks developed in both horizontal and vertical planes. When the furnace was next brought into operation the cracks became filled with liquid matte which solidified as it ran into the lower and cooler parts of the bottom. As the furnace reached full operating temperature and full expansion, growth of the bottom occurred. Repetition of these cycles resulted in a bottom permeated with copper in the form of matte filled fissures. Eventually, the furnace bottom consisted of nothing but pieces of slag surrounded by matte. As bottom temperatures increased the pieces of slag were attacked on every face by matte.

History of Furnace

No. 4 furnace was first put into operation in April 1922. The original bottom was built on a poured slag base 5 ft thick and a layer of crushed silica, from 30 to 36 in. thick, was tamped in on the slag base and fused in the conventional manner.

Shortly after the furnace was fired an undetermined amount of the silica bottom came up. In November 1924 the furnace was shut down after 682 days of operation. Drilling of the bottom showed that the silica bottom had been replaced, to a depth of 6 to 10 in., by a heavy slag of high magnetite content, containing 17.8 pct copper. This ferrous slag had built up to a level of 6 to 10 in. above the top level of the original silica bottom.

Although more than half of the silica bottom remained in place at the time of the 1924 drilling, the remaining part had been enriched to an average copper content of 21.6 pct. The slag base underneath had been enriched slightly, to 2.05 pct copper.

H. FOARD is Superintendent, Miami Plant, International Smelting & Refining Co., Miami, Ariz. This paper was presented at the AIME Annual Meeting, Chicago, Feb. 14 to 17, 1955.

The furnace bottom inventory indicated by the 1924 drilling showed a total of 543,128 lb of copper, of which 433,492 lb was below floor level. Fig. 2 diagrams the results of the 1924 drill survey.

War production and increasing labor costs necessitated a gradual increase in daily tonnages after 1944. The advent of natural gas fuel in 1946 allowed greater flexibility of operation but the lower flame temperature of gas increased the difficulty of maintaining tonnage rates at a high level. In addition, changes in orebodies of the district caused an increase in the slimes content of concentrates received and resulted in a very wet and refractory charge. For months the furnace was on a charge of over 15 pct moisture. Maintenance of high operating rates required greatly increased fuel consumption of accelerated rates of heat release with attendant deterioration of the bottom. Table I gives the pertinent operational data for No. 4 furnace.

Removal of Furnace Bottom

Operations were transferred to another furnace at the end of 1953, and No. 4 was shut down for rehabilitation. The furnace was skimmed as well as possible and matte tapped until flow through the taphole ceased. While the furnace was still under fire, berms were thrown up to form a sump of the entire converter pit floor (elevation 3551 ft or 11 ft below the furnace floor elevation of 3562 ft).

A 4-in. drifter was set up 35 ft from the end of the furnace. Round steel 1¼ in. diam, in coupled lengths, was chucked in the machine and the long length of steel supported at intervals on tripods. A 4 in. tungsten carbide insert bit was used, and the hole was started through the end foundation wall 30 in. above the converter pit floor level.

Air was fed through the water feed of the machine to remove drill cuttings. In addition a small amount of water was fed into the same line, with

Table I. Pertinent Operating Data for No. 4 Furnace

Date	Furnace Days Operated	Total Tons Charge	Avg Tons Per Day	Million BTU Per Furnace Day
Oil fuel — calcine charge	1265	665,787	526	2168.5
Oil fuel — wet charge	2500	1,333,501	536	2936.7
Gas fuel — wet charge	1908	1,127,486	591	2603.3
Total	5681	3,126,864	550	2969.5

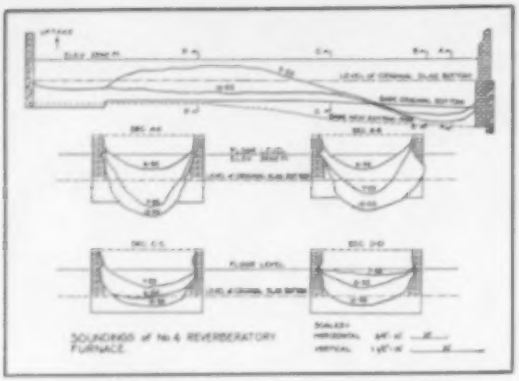


Fig. 1—Soundings of No. 4 reverberatory furnace in December 1953 demonstrated a pool of molten matte 8 ft in depth at a point 20 ft in front of the bridge wall.

the idea of using just enough water to cool the bit by vaporization while avoiding the presence of liquid water in the hole. Drilling, although slow because of the large bit, proceeded satisfactorily to a depth of 10 ft. At that point the bit failed, probably because of a stoppage in the water feed.

After the failure of the bit, a ⅜-in. pipe with 100 lb air supply was pushed through a port in the arch to the bottom of the matte pool, and the matte was blown vigorously. At the same time extension of the drill hole by means of the oxygen lance was in progress. The lance was successful in extending the hole 10 ft to tap the matte pool. Flow started 6 hr after drilling had begun and continued for 10 hr. Matte drained from the furnace was estimated at 1200 tons, containing 43 pct copper.

After a short period of cooling, the bridge wall and its foundation wall were removed, and the entire furnace bottom was blasted and slushed to the converter pit. The cost of drilling, blasting, and slushing amounted to $3.00 per ton. Approximately 2500 tons of furnace bottom was reclaimed. The analysis of the reclaimed furnace bottom is shown in Table II.

Material removed from the bottom was of a heterogeneous nature. Segregations of crystallized magnetite of 2 to 3 cu ft in volume were observed, especially at various points along the walls. Flecks of metallic copper were found throughout the bottom and, at a point near a front corner a ton or two of the old silica bottom, heavily enriched with copper, was encountered. The entire bottom was heavily fissured and altered, with no trace remaining of its original characteristics. Major matte concentrations were found in the following locations:

A—A zone beginning 5 ft from the bridge wall, 25 ft in length. Two thirds of this 25x25 ft zone showed matte penetration to a depth of 2 ft into the clay underlying the poured slag base, or 11 ft 1 in. below the furnace floor level.

Table II. Chemical Analysis of Reclaimed Furnace Bottom

Element	Pct
Cu	12.74
SiO_2	23.1
Al_2O_3	6.5
Fe	35.0
CaO	2.3
S	0.3

B—A zone 35 ft in length beginning 50 ft from the bridge wall. This zone extended from wall to wall, and at places well into the wall. Penetration into the underlying clay was 2 ft.

Matte penetration was found to have extended entirely through the brick beneath the bridge wall and had been contained in places only by the concrete foundation wall. Only careful attention to fettling during the last few years of operation had prevented a disastrous runaway.

Heavy sidewall wear extended 60 ft from the bridge wall, in an area which had been considered to be fully protected by the furnace charge. Wear occurred in the form of a deep gouge at the matte line. At one point, 25 ft from the bridge wall and 18 in. below the floor line matte had cut away half of the bottom end of a buckstay.

Installation of New Bottom

After removal of the old bottom the furnace walls and buckstays were repaired. Most of the old arch was held in place, to permit firing the furnace during bottom replacement. Following completion of wall and flue repairs the furnace was put under fire.

Four vertical burners inserted through the arch along the center line were used. Pouring of the slag bottom was started on May 10, 1954, at which time the bottom had reached a temperature of 1650°F.

Table III. Temperatures Recorded at Thermocouples, °F

Thermocouple	May 10*	May 24**	June 24	July 24	Sept 21	Oct 1***
No. 1	581	1310	1085	806	500	482
No. 2	342	1211	860	606	374	356
No. 3	122	869	464	326	212	194

* Pouring started ** Bottom completed *** Operation started

Approximately 2500 tons of molten reverberatory slag were poured through the converter slag launder to complete the new bottom. The pouring required slightly more than 7 days, during which time the temperature of the slag surface was held at 2000°F, as measured by optical pyrometer. At that temperature, although a very thin surface crust existed, slag flowed freely from one end of the furnace to the other beneath the crust.

Following the pouring of the slag bottom 100 tons of flotation tailings were puddled in by means of a natural gas lance. The tailings were charged in bulk along the sidewall and mixture into the bath

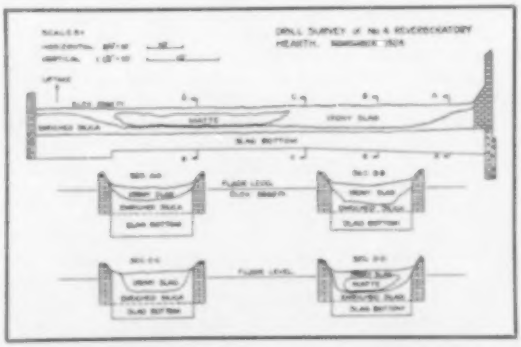


Fig. 2—Drill survey of No. 4 reverberatory furnace made in November 1924 after two years of operation showed that more than half of the silica bottom remained in place.

Table IV. Chemical Analyses of Slag Bottom as Poured and Tailings, Pct

Material	Cu	SiO_2	Al_2O_3	Fe (Total)	CaO	MgO	Fe_2O_3
Slag	0.52	35.6	3.0	34.9	5.7	1.1	7.9
Tailings	0.02	72.0	13.4	2.5	0.6	0.8	—

was carried on gradually over a period of three days. They were mixed into the slag thoroughly and formed a mushy layer over the entire surface. The temperature of the bath was then raised to 2400°F and firing at that temperature continued for 24 hr, without agitation. This final firing caused the formation of a surface which was infusible at 2400°F.

During the rebuilding of the furnace wall a thermocouple well was installed in the sidewall, approximately 25 ft from the bridge wall, at an angle of 35° from the vertical. The bottom end of the well was 1 ft inside of the furnace wall and at the bottom of the slag fill, or 7 ft 6 in. below the floor line. An iron-constantan couple was placed at the bottom of the well, designated as couple No. 1. Two similar couples were installed in the well above No. 1 couple, No. 2 being 3 ft, and No. 3 6 ft from No. 1 as shown in Fig. 3. Temperatures noted at the couples after May 23, 1954, on which date firing of the bottom was completed and burners removed are in Table III.

The old furnace arch was repaired during the period allowed for cooling of the bottom, and the furnace was put into operation on Oct. 1, 1954. Soundings of the bottom since that date indicate that, although the silica-enriched top of the slag bottom did not prove as resistant to erosion as had been hoped, it will perform satisfactorily. After two months of continuous operation the bottom was eroded to a depth of 22 in. in a localized zone of maximum wear, 12 ft in front of the bridge wall. Throughout the furnace generally, bottom erosion amounted to an average of 9 in. This wear is not excessive when it is considered that the furnace was operated at the very heavy gas flow of 200,000 cu ft per hr, or a heat release of 1,700,000 Btu per sq ft of combustion area per hr.

Erosion of the bottom shows no further progression, and it is believed that reasonable attention to fettling will keep it in check.

Composition of Slag Bottom

The pouring of a solid slag bottom, instead of fusing in a silica bottom on poured slag, was based on plant records. Those records showed that, especially in the region of converter slag return, silica bottoms eroded rapidly. In addition, the entire silica bottom showed high copper absorption. After some years of operation the silica bottoms disappeared and operations continued successfully on the remaining slag bottom. Failure of the slag bottom seemed to stem from the filling of fissures with matte rather than gradual erosion of the slag surface.

Table V. Analyses of Furnace Bottom After Firing, Pct

Top	Cu	SiO_2	Al_2O_3	Fe	CaO	MgO
0 to 4 in.	0.56	31.9	6.1	21.5	8.0	1.2
4 to 16 in.	0.19	39.9	6.1	31.7	6.7	1.3
16 to 28 in.	0.19	40.0	6.1	31.5	7.3	1.3

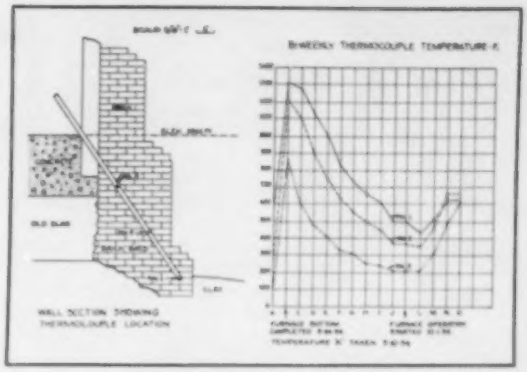


Fig. 3—Iron-constantan thermocouples were installed in the side wall. No. 1 couple was 1 ft inside furnace wall and at bottom of slag fill. No. 2 and 3 couples were 3 and 6 ft respectively from No. 1.

Reverberatory slag produced at the time of pouring the furnace bottom was very fusible, and the admixture of other ingredients after pouring, to raise the fusion point was desirable. Flotation tailings were available at minimum expense and were deemed suitable for addition.

Analyses of the slag bottom as poured and of the flotation tailings are given in Table IV. The composite furnace bottom was cooled after firing, and samples taken for chemical and mineralogical analysis. Chemical analyses are shown in Table V.

The top 4 in. of the completed bottom contained much quartz which, in part, appeared to retain the original grain outlines as present in the tailings. The specimen was estimated to contain 30 pct quartz, 30 pct glass, 15 pct aegirinaugite sheafs, and 25 pct microlites. Segregations of coarse aegirinaugite-magnetite spherulites were found. X-ray patterns did not show fayalite to be present.

Various sections within the range of 4 to 16 in. were examined. No chemical nor mineralogical differences of importance were found between the various levels of this range, although texture was slightly coarser in the lower levels. Secimens were estimated to contain, by volume, 30 pct fayalite, 60 pct aegirinaugite, 9 pct magnetite, and 1 pct metallic iron. Intergrowth of alternate plates of fayalite and aegirinaugite furnish the dominant texture. Individual plates of augite varied from 0.05 mm thick at the top of the 4 to 16 in. range to 0.16 to 0.25 mm at the bottom. Individual plates of fayalite are from 0.03 mm to 0.8 to 0.16 mm thick, at corresponding levels.

Acknowledgments

Appreciation is due to officials of the International Smelting & Refining Co. for their permission to publish this paper. Acknowledgment is also made to P. D. I. Honeyman, general manager of the Miami plant, whose work in 1924 furnished information regarding the furnace bottom at that time.

Thanks are due to the geological dept. of the Anaconda Co., and especially to M. H. Gidel and Paul L. Cloke, for the examination of specimens at the geological research laboratory.

A Meeting
At the Market Place-

Coal Looms As Low Cost Power Source For Aluminum Production

by Arthur F. Johnson

PLANT sites for the light metal industry must be located where ample low cost power is available. In the first half of this century, hydropower has been the only source of low cost power. Now the best of the hydro sites have been exploited and the remaining do not promise enough water to yield capacity operation for the power plant all the year around. In searching for economic plant sites, engineers have concentrated upon locations where vast water power potential could be developed.

The Olin Mathieson Chemical Corp., like many others, has been studying the economics of developing water power to make aluminum. Likewise, natural gas, lignite coal, and even atomic energy have been considered as possible sources of power to make aluminum. In these economic studies, consideration has been given not merely to investment and production costs, but to total costs of delivering fabricated products to the market center of the U. S.

When *delivered costs* are considered and full accounting is taken of the newest techniques of mining bituminous coal and utilizing it to make electric power, one arrives at the inescapable conclusion that coal has emerged as the economic answer to American aluminum production. Also coal should be considered as the logical new source of firm power for all major electro processes in the U. S.

Low Cost with Integration

Considering both investment and operating costs, it is estimated that power from coal can be made for as little as 3.5 mills per kw hr in plants of 300,000 kw capacity or more when made firm 365 days per year by interconnection with a public utility system. In many cases it will pay to have a utility company build and operate the power plant as well as firm up the load when generators are shut down for repairs. Power from coal can cost less than new power from natural gas, hydro development or atomic energy. Moreover, power from coal is available near the market center of the U. S. Two year reserves of cheap natural gas are chiefly limited to the Gulf Coast, and the remaining great hydroelectric potential is in the northeast U. S. or Canada —both of which are too far removed from raw materials and from U. S. manufacturing and consuming centers in the east and midwest.

By integrating coal mines with power plants and industrial plants, lowest costs are secured. The integrated plants, if located on a navigable river, enjoy water compelled freight rates on raw materials. Integrated plants can be economic even when built on much smaller streams if cooling towers or a cooling reservoir is used for condensing steam. The coal mining areas have ample rural labor available and enjoy some tax advantages.

Considering all these favorable factors, it will pay all large power consuming industries to study the economics of integrating their plants with power plants located at coal mines. Coal mining states should examine not only the means by which low cost power can be made available to industry but also all the other factors which industry must consider in selecting a plant site involving a substantial investment. The potential business that might be captured by these states in the next 20 years is not just an estimated 66 million tons more of coal

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Table I. Total Fuel and Operating Costs of Texas Power & Light Co., 1939

Costs	Mills per Kw hr
Fuel (lignite coal mined underground)	0.94
Operating labor	0.30
Misc. operating expense	0.04
Boiler maintenance	0.11
Other maintenance	0.09
Total	1.48

needed to make power but also the metal and chemical businesses worth billions of dollars per year.

More Power From Less Coal

In 1938, the coal burning plants of the electric utility industry required an average of 1.4 lb per kw hr. By 1954 this average had been reduced down to about 1 lb and there were at least 34 plants that required less than a pound of coal per kw hr. With a modern boiler (operating at 1800 lb pressure and 1000° to 1050°F) and a modern turbine and generator, it is possible to generate 1 kw hr with 9300 Btu's of fuel.[1] Depending on the mining district, 1 lb of coal contains about 10,000 to 14,500 Btu's. The yearly fuel bill for any large central station boiler today is usually greater than the installed cost of the boiler equipment. Therefore, any increase in cycle efficiency represents a worthwhile amount of money.[2] One of the latest advances in steam boiler design is that incorporated in the Philo plant of the Ohio Power Co. of the American Gas & Electric system. Here the net plant heat rate is expected to be about 8500 Btu's per net kw hr depending on the vacuum obtained in the condenser. In the month of November 1954 a heat rate of 9050 was actually achieved in the Muskingum plant of this company. The efficiency trend of coal utilization since the year 1900 and projected to 1956 is shown in Fig. 1.

In the 1925 to 1930 construction era, it took about one power plant operating employee per 1000 kw of installed capacity. By 1951 a two unit, 325,000 kw coal burning power plant used only 0.31 employees per 1000 kw capacity.

More Coal With Fewer Men

Since 1905 coal production per man per day in the average U. S. mine has increased from 3.5 to 8.4 tons as shown in Fig. 2. However, such statistics do not nearly reflect the great increase in efficiency. For example, some efficient new mines are producing more than 25 tons per man day underground.

Coal mining costs are being cut by the use of such improvements as the Armstrong coal breaker, mechanical cutting, and extensible conveying from the coal mine face to the power plant boilers. The Armstrong breaker uses highly compressed air instead of dynamite to blast down the coal and so eliminates disagreeable fumes. The coal miner of today is the operator of huge undercutting and loading machines. He receives the highest rate per hr in the mineral industry plus a pension based on 40¢ per ton of coal mined, but his job is intermittent. The coal miner and the coal community need stable industries with steady employment 12 months per year. If there were more power-consuming heavy industries located at the coal mines, every person in the mining community could benefit from ultimate utilization of the coal rather than on the mining alone. This would insure a permanent healthy industrial growth in the coal mining states. This is the magic formula for creating boom towns instead of ghost towns.

Costs of Power from Coal

The costs of electric power generation from coal are broadly made up of fuel cost, other operating costs, and money costs consisting of interest, amortization, taxes, and insurance on the investment. On large plants built today money costs may be kept down to as low as 1.0 mill. Accordingly, to attain a 3.5 mills total power cost, fuel plus operating costs should be not more than 2.5 mills. Total fuel and operating costs of only 1.48 mills per kw hr were attained by the Texas Power & Light Co. as early as 1939 when their costs were as shown in Table I.

To attain such low costs TP&L integrated a coal fired steam plant with a coal mine and used a reservoir for cooling the condensers. The lignite coal was mined underground.

TP&L recently completed for the Aluminum Co. of America a 240,000 kw power plant and a reservoir for cooling adjacent to Alcoa's Rockdale, Texas smelting plant. The power plant uses lignite fuel from a nearby deposit. The mine, the power plant and the aluminum manufacturing plant add up to a wonderful asset for the state of Texas. Now what about the vastly greater possibilities for cheap

Table II. Comparison of Estimated Costs to Make Aluminum of Hydroelectric Power in British Columbia with Coal Fired Steam Electric Power in Southern Illinois[3,4]

	British Columbia Hydroelectric Plant			Coal Fired Power Plant Southern Illinois
	First Stage	Second Stage	Third Stage	
Installed Kw	366,000	732,000	1,464,000	300,000
OPERATING COST (mills per useful Kw hr)				
Plant labor, supt. & repair & maint.	0.25	0.25	0.25	Coal 0.50
Tunnel—R & M	0.02	0.01	0.01	1.21
Dam—R & M	0.03	0.01	0.01	—
Transmit line—R & M	0.03	0.01	0.01	—
Land rental	0.02	0.00+	0.00+	Taxes 0.19
Water rental	0.07	0.06	0.06	Ins 0.00+
Total	0.42	0.34	0.34	1.90
MONEY COST				
Investment (millions of dollars)	165	226.3	390.2	39.6
Average interest 1st 20 yr	@ 4 pct 2.70	@ 4 pct 1.22	@ 4 pct 1.06	@ 5 pct 0.67
Normal amortization	2.21*	1.01*	0.87*	0.52**
Total	4.91	2.23	1.93	1.19
TOTAL, All Plant Costs	5.33	2.57	2.27	3.09

* 40 yr. ** 35.7 yr.

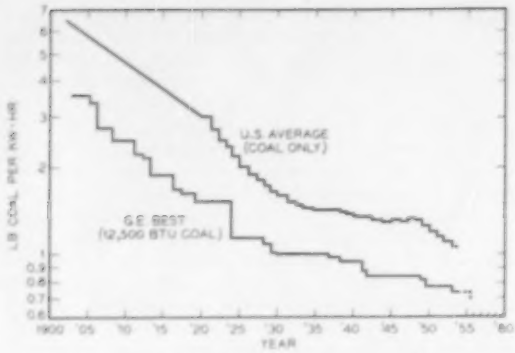

Fig. 1—Efficiency trend of coal utilization since the year 1900 and projected to 1956 is shown. Graph is adapted from data supplied by the General Electric Co.

power from dependent industries that could be located at coal mines in Illinois, Indiana, Kentucky, Ohio, Pennsylvania, West Virginia, Virginia, Tennessee, and Alabama? These states have almost unlimited reserves of cheap coal located near the manufacturing and consuming centers of the U. S.

Each industry and company has a particular set of plant site requirements that may limit choice of economic sites to a few localities. However, all sites in these states have the common advantage of nearness to market. It is assumed that any site in the eastern and midwestern states is close enough to the U. S. market so the average customer can be reached by an overnight haul.

That coal power is cheaper than hydropower is revealed by Table II. It compares the estimated cost of the new 366,000 kw and projected 732,000 kw and 1,464,000 kw hydro plants in British Columbia, Canada, with a 300,000 kw coal fired steam plant in southern Illinois. This comparison is chosen because it probably represents the lowest cost hydropower that can be developed in western Canada or Alaska. The Canadian project has been given the advantage of a 4 pct interest rate compounded with a 5 pct interest rate on an investment in southern Illinois. It is easy to see that the huge investment cost makes hydropower plants uneconomical compared to efficient coal fired steam power plants of comparable size. In the above examples power is only 3.09 mills per kw hr on the coal fired plant in Illinois compared with 5.33 mills in Canada in a plant of comparable size. Although the huge 732,000 kw plant in Canada may later produce power for 2.57 mills, the freight on raw material to British Columbia and finished product back to the U. S. markets and import duties will far over-balance the slightly lower power cost in Canada. In fact, if aluminum is produced in British Columbia the freight to U. S. market centers plus import duty will about equal the total cost of power.

A great deal of publicity has been given the possibilities of producing cheap power from atomic energy. From the best information now at hand, it seems that nuclear power would cost about twice conventional power in areas where the price of coal or oil is neither excessively high nor low. The cost of atomic power can conceivably be modified somewhat by the sale of material products and by-products. In 15 or 20 years atomic power may become competitive but today coal power is cheaper.

The coal fired steam plant with electrostatic precipitators to clean ash particles from the stack gases is one of the cleanest as well as most efficient plants which industry has ever designed. It is to the credit of the power industry that it has developed a plant that can operate in the heart of a great city with minimum nuisance to its inhabitants.

Table III. Utilization of Nordberg Gas Burning Radial Engines for Electrolytic Reduction of Aluminum*

Costs	One Pot Line (45,900 kw)	Two Pot Lines (91,800 kw)	Three Pot Lines (137,700 kw)
Estimated Operating Costs, Mills per Kw hr			
Fuel, 10¢ gas	1.0500	1.0500	1.0500
Lub oil, 40¢ bbl	0.1332	0.1332	0.1332
Maintenance	0.2690	0.2690	0.2690
Labor	0.3872	0.3146	0.2914
Misc.	0.0042	0.0042	0.0042
Total	1.8436	1.7712	1.7478
Total Operating Costs, Mills per Kw hr*			
10¢ gas	3.47	3.40	3.38
15¢ gas	4.09	3.83	3.80
20¢ gas	4.82	4.45	4.43

* Total calculated by adding to estimated operating costs the money cost and taxes amounting to 1.63 mills per kw hr (5 pct amortization, 2½ pct interest and 1 pct taxes; total 8.5 pct of $168 cost per firm kw).

Natural gas diesel engines used for aluminum reduction plants have the advantage of being able to produce dc rather than ac power. This saves about 6 pct power loss in conversion from ac to dc in transformers and rectifiers plus the investment and operating costs of the rectifier station. The money cost plus operating cost of converting ac to dc amounts to about 0.4 to 0.5 mills per kw hr.

Table IV. Present and Projected Power Requirements and Tonnage Expansion for Selected Industries[45]

Industry	Power for Reduction Kw-hr per Ton	Estimated Industry Expansion, 1954 to 1975, Tons	Estimated Firm Expansion, 1954 to 1975, Kw
Titanium*	40,000	500,000	2,280,000**
Steel	325	50,000,000	3,000,000**
Aluminum	18,000	2,000,000	3,083,000***

* Overall power requirement for titanium includes power consumed by production of magnesium used to reduce titanium.

** Power consumption in first column is multiplied by tonnage in second column divided by 8760 kw hr per yr of firm power.

*** Estimated as above excepting 500,000 tons scheduled Kitimat third stage deducted from second column.

Cheap electric power is being made with diesel engines to produce aluminum on the Gulf Coast. Table III shows the cost of power that can be produced with natural gas costing 10¢ per million Btu's.

Present prices for gas on the Gulf Coast are in the 15 to 20¢ range of cost. The field of the natural gas engine for making new cheap power still looks important but coal is competitive and provides power nearer the market centers of the U. S. A.

Comparing power generation cost by new hydro development, natural gas diesel engines, and coal, Tables II and III, reveal coal is the cheapest power. In the 25 years from 1925 to 1950 power consumption in the major electroprocess industries increased 815 pct. From 1950 to 1975 an expansion of 452 pct from 37.5 billion kw hr to 207 billion kw hr is confidently predicted.

It can be deduced from present and projected electric energy consumption that the power consumption in major electroprocesses will require new power plants to provide 20 million additional firm kw by 1975. If all of this power were provided by coal it would take 66 million tons at 0.75 lb per kw hr. This is a significant amount since the Paley Re-

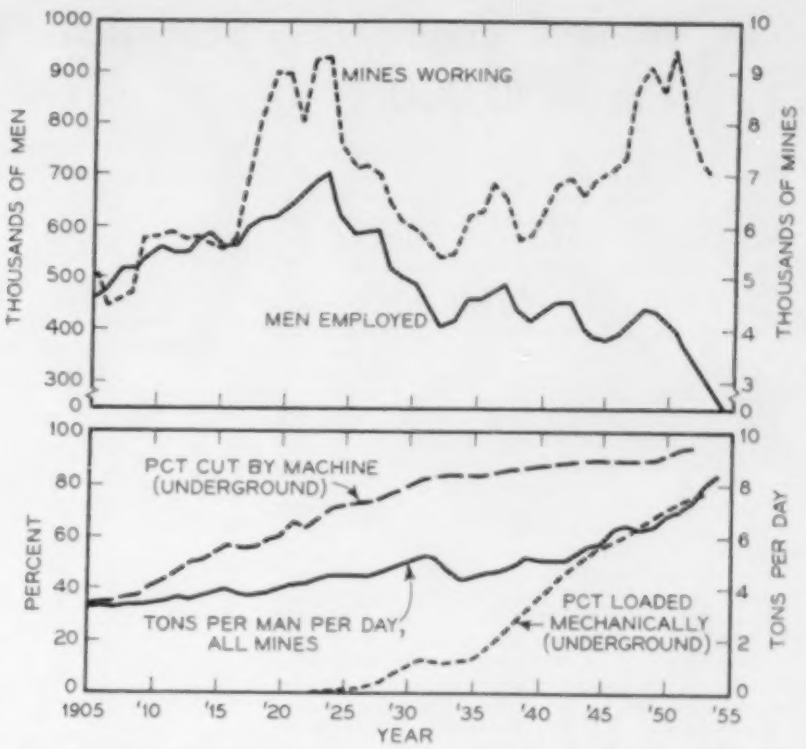

Fig. 2—Trends of employment, mechanization, and output per man at bituminous coal and lignite mines in the U. S. are plotted. Graph reproduced from the U. S. Bureau of Mines Minerals Yearbook.

port shows that in 1950, 115 million tons of coal were used to produce electricity in the U. S. A.

The Paley Report also shows power requirements for selected process industries. Table IV shows power requirements estimated for selected process industries per ton of product and for U. S. industry in 1975. By the year 1975 millions of firm kw will be needed to make the additional needs of the U. S. for aluminum and titanium. It also appears that electric furnace steel expansion may be more economic than open hearth steel expansion if low cost, 3.5 mills per kw hr, power is made available.

Suggested Governmental Action

Slight tax advantages are given to new industries in some states. In areas where a great surplus of labor exists rapid amortization of new facilities is allowed up to 100 pct of the value under the Federal tax laws. On projects of the same kind in other areas only 85 pct is allowed. Under the new tax laws such advantage of establishing in distressed labor areas is slight. Considering the payroll and tax income which the states would gain from establishing new industries, it would doubtless pay them to allow some further tax incentive to new industry while such industries are amortizing their plants.

In this atomic and H bomb age the Federal government is showing increased interest in dispersing production facilities, particularly those which are associated with essential civilian or defense purposes. In general, such dispersal tends to move facilities into areas of labor unemployment. In addition to the tax advantages offered by rapid amortization it might very well be appropriate for the government to consider additional incentives for such dispersion. For example, might it not be feasible for the government to encourage dispersion by offering government guarantees on all or part of private loans in a manner similar to their reinsurance of FHA loans which have done a good bit to expand home construction?

Considering all these factors it is apparent that power plants integrated with coal mines are the key to low cost industrial power particularly if located near consuming areas. Utilization of the huge coal potential now will have great social as well as economic advantages to the people of the U. S.

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Mineral Engineering Student Enrollment Passes 10,000

by William B. Plank

FOR the first time in four years the number of mineral engineering students in the 228 engineering colleges of the U. S. and Canada exceeds 10,000. According to figures just released by the American Society for Engineering Education, the current enrollment of all engineering students is 222,178.

The total of 10,274 mineral engineering students, undergraduate and graduate, is a 4½ pct increase over a year ago. Total engineering is 11 pct larger than last year, about the same as in 1949-50, and 24 pct above 1950-51.

Except for the metallurgical engineers, fewer by 3.2 pct than last year, mineral engineering students have increased. Ceramic engineers, with 626, lead the list with a 12 pct increase. The increases in the other branches are: geological, 10 pct; petroleum, 9 pct; and mining, 3 pct.

Among the 23,272 first degrees granted last year by the schools in the U. S. and Canada, 1476 were to mineral engineers. Of these, 231 were in mining, 486 in metallurgy, 460 in petroleum, 113 in ceramics, and 186 in geological engineering.

Looking ahead, an increasing number of graduates from U. S. mineral engineering schools may be expected for the next four years—except in metallurgical engineering. This is shown by the increased sizes of the four-year class groups. In metallurgical engineering, however, the entering class is 9 pct smaller than last year's entering class—the smallest, in fact, of the four classes in that branch. Additional precollege guidance, or scholarship aid, is indicated for this group.

This year 1179 women were enrolled in the engineering schools of both countries, an increase of 26 pct. Last June 36 women received their degrees in engineering, but according to a recent NEA release, many more women engineers are needed in government service and in industry.

Although current growth in the entire engineering group is encouraging to industry, it is yet not as rapid as in the other college groups. Of 2,499,750 U. S. college students, only 7.5 pct are in engineering as compared with 10.5 pct in 1947-48.

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First Degrees, All Engineers, U. S. and Canada

1942-43	14,714	1949-50	54,441
1943-44	11,155	1950-51	43,762
1944-45	4,724	1951-52	31,660
1945-46	8,219	1952-53	25,332
1946-47	19,272	1953-54	23,272
1947-48	30,018	1954-55	28,416
1948-49	46,934		

Engineering Student Enrollment Comparison Undergraduates and Graduates

Courses	1949-50	1950-51	1951-52	1952-53	1953-54	1954-55
Mining	2,223	1,607	1,339	1,168	1,089	1,118
Metallurgical	3,887	3,511	2,996	3,085	3,159	3,050
Petroleum	4,175	3,321	2,699	3,448	3,772	4,110
Ceramic	998	886	667	579	557	626
Geological	?	1,349	1,027	1,173	1,249	1,370
Total Mineral Engineers, U.S.	11,283	10,674	8,727	9,453	9,826	10,274
Chemical	10,976	16,323	14,079	15,026	15,858	17,097
Civil	29,109	24,505	21,801	22,393	23,075	23,799
Mechanical	45,632	35,347	30,420	32,791	35,097	38,587
Electrical	45,711	34,105	29,966	32,848	37,633	43,305
Industrial	?	?	6,578	6,643	6,786	7,168
Other Engrs.	68,001	59,308	36,123	39,364	43,557	47,224
Total Engrs., ECPD			147,694	158,518	171,832	187,454
Other U. S. Engrs.			17,943	18,031	21,501	26,960
Total U. S. Engrs.	219,712	180,262	165,637	176,549	193,333	214,414
Canadian Engrs.	6,821	4,285	5,433	5,812	6,894	7,764
Total Engrs. U. S. and Canada	226,533	184,547	171,070	182,361	200,227	222,178

Graduate Engineering Enrollment, 1954-55, U. S. and Canada

			Degrees, 1953-54	
Courses	Masters	Doctors	M.S.	Ph.D.
Mining	64	9	23	3
Metallurgical	483	243	139	53
Petroleum	165	15	59	6
Ceramic	77	34	34	16
Geological	98	30	9	2
Total Mineral Engrs., U. S., ECPD	897	331	263	80
Chemical	1,725	622	448	133
Civil	1,900	339	563	43
Mechanical	3,040	421	710	72
Electrical	5,426	892	978	111
Industrial	1,487	91	367	6
Other Engrs.	1,987	583	732	145
Unclassified	610	0	69	0
Total for ECPD Schools	17,072	3,279	4,130	590
Other U. S. Schools	369	2	47	0
Total for all U. S. Schools	17,441	3,281	4,177	590
Canadian Schools	167	68	90	23
Grand Total	17,608	3,349	4,267	613

Undergraduate and Graduate Engineering Enrollment, U. S. and Canada, 1954-55, 228 Schools

Schools	Courses	Frosh.	Soph.	Junior	Senior	5th Yr. & Others	Total Undergrads	Grad. Students	Grand Total
31	Mining	281	276	240	226	20	1,045	73	1,118
49	Metallurgical	651	586	502	476	239	2,314	736	3,050
22	Pet. & N. Gas	1,291	1,003	827	674	135	3,930	180	4,110
14	Ceramic	150	145	113	90	17	515	111	626
25	Geological	445	301	253	228	15	1,242	128	1,370
	Total Mineral Engineers U. S.	2,618	2,311	1,935	1,696	426	9,046	1,228	10,274
107	Chemical	4,503	4,075	2,986	2,180	1,006	14,750	2,347	17,097
137	Civil	5,816	5,403	4,406	3,982	1,943	21,560	2,239	23,799
133	Mechanical	9,215	9,089	7,077	5,576	4,169	35,126	3,461	38,587
138	Electrical	10,381	9,938	7,373	5,035	4,260	36,987	6,318	43,305
65	Industrial	1,052	1,316	1,361	1,348	513	5,590	1,578	7,168
	Other Engrs.	5,134	4,228	3,379	2,834	1,237	16,812	2,570	19,382
94	Unclassified	16,812	1,985	122	16	8,297	27,232	610	27,842
150	Total Engrs. ECPD Schools	55,531	38,345	28,690	22,677	21,851	167,103	20,351	187,454
68	Other U. S. Engr. Schools	9,974	5,729	3,642	2,879	4,374	26,589	371	26,960
218	Total U. S. Engineers	65,505	44,074	32,341	25,547	26,225	193,692	20,722	214,414
10	Canadian Engineers	2,520	1,941	1,477	1,239	453	7,529	235	7,764
228	Grand Total U. S. & Canada	68,025	45,915	33,818	26,786	26,677	201,221	20,957	222,178

Fluidity spiral ½ in. square shows 27 in. flow with good surface quality.

Large Titanium Castings Produced Successfully

by R. A. Beall, F. W. Wood, and A. H. Roberson

SUCCESSFUL castings weighing as much as 110 lb have been made by consumable electrode arc melting and casting in vacuum. The melt is contained in a water cooled copper crucible. Bottom and side pour crucibles were used successfully.

The maintenance of deep pools of molten metal with enough superheat to produce good metal flow depends on a maximum melting rate which is realized when the vacuum is maintained just short of the point where glow discharge occurs.

Ability to cast significant amounts of titanium metal successfully requires at least two conditions: A—a large volume of molten metal and, B—enough superheat to provide fluidity. Consumable electrode arc melting meets both of these demands.

The amount of metal available for casting depends, of course, on the rate at which the melting is done. Experience with both zirconium and titanium has pointed to the fact that melting in a vacuum greatly increases the melting rate; the maximum being realized at a point just above the glow discharge pressure of about 30 mm of mercury.

For large pools the power input frequently reaches 9000 amp, which will melt as much as 9 lb of titanium per min. Superheat is a function of power input and is essential for the production of sound castings.

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The use of water cooled copper crucibles to contain the melt in effect results in the melt being held in a thin-walled titanium crucible. The same condition may be realized when the container is made from graphite or other materials. However, when the charge is superheated in the other containers there is a distinct possibility of a crucible-melt reaction occurring.

Castings

Several sizable castings have been produced by the bottom pour technique. Fig. 1 shows five 75-mm breech liners weighing about 20 lb each, as well as several smaller shapes. Fig. 2 shows a titanium breech block weighing about 75 lb. The total pour, including gates and risers weighed 110 lb. The 14x14x1 in. plate and riser shown in Fig. 3 is one of several similar castings in which about 65 lb of metal was poured each time.

Fluidity spirals ½ in. square cast with the pour-over-lip equipment indicated that up to 27 in. flow might be expected with good surface quality.

The surface appearance of the castings was good and only a few folds or imperfections were observed. Ultrasonic and radiographic inspection revealed the presence of a few gas pockets or voids. It should be emphasized that only a minor amount of attention was given to mold design or the placing of gates and risers. If special attention had been diverted to this feature, it is believed that most of these defects could have been obviated.

Research on Castings

Considerable research has been done on the casting problem, some of which has been successful.

Fig. 1—Various shapes including five 75-mm breech liners weighing about 20 lb each were cast by the bottom pour technique.

Fig. 3—Similar castings as the above 14x14x1 in. plate and riser were cast. In each about 65 lb of metal was poured.

Kuhn[1] was the first to make carbon free finished mechanical testing specimens, including the threads, by casting. Kroll and Gilbert[2] of the U. S. Bureau of Mines reported a method for casting zirconium that was reportedly useful for titanium. A side pouring furnace was developed by Lang, Kura, and Jackson,[3] at Battelle Memorial Institute and Simmons and McCurdy[4] reported success with a bottom pour type of casting furnace. According to Arnold, Malone, Clark, and Finlay,[5] there is a sound metallurgical basis for belief that titanium castings can be produced on an industrial scale. The work at Rem-Cru Titanium Corp. confirmed this work.

Contrary to general opinion of two years ago, it is now generally accepted that titanium metal must be essentially carbon free. This change of view has rendered some of the early casting equipment obsolete. Furthermore it has limited the capacity of other equipment, since dependence was placed on the use of graphite crucibles and/or graphite arc

Fig. 2—Titanium breech block weighing 75 lb was cast. Total pour, including gates and risers, weighed 110 lb.

Fig. 4—Two electrodes, one made from scrap plate and the other from reclaimed casting scrap, are shown.

electrodes. Although low carbon melts can be prepared in graphite crucibles, very close control of applied heat must be maintained. If experience in other casting fields applies, superheated metal must be obtained to yield good quality castings. This makes even more difficult the problem of producing carbon free metal. During the past few years wide experience has been gained at the U. S. Bureau of Mines Albany Station on consumable electrode arc melting of zirconium. Recently an extension of this work[6] has shown that exceptionally deep pools of superheated metal could be produced, and it seemed that the pouring of substantial quantities of metal could be feasible. The marked similarity of titanium and zirconium pointed to the possibility that titanium also might be poured in significant quantities. Preliminary work confirmed the possibility, for which patents have been applied.[7]

Materials

Two types of melting stock were supplied—titanium sponge and scrap plate stock containing 8 pct Mn.

The sponge was briquetted to 2-in. square compacts, 20 in. long and consumably arc melted into 6-in. diam ingots. This produced an electrode that would supply enough metal for large scale remelting and pouring. Scrap plate stock was arc welded in an inert atmosphere to provide another sizable electrode. Fig. 4 shows two electrodes, one made from scrap plate and the other from reclaimed casting scrap.

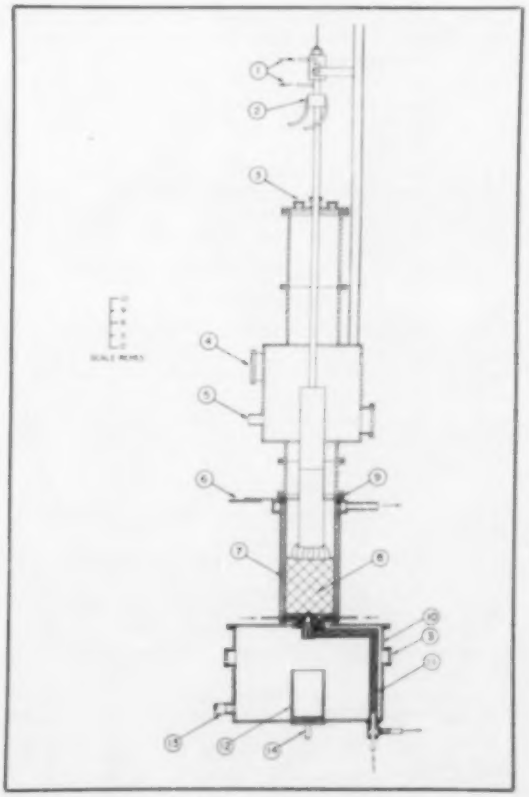

Fig. 5—Bottom pour furnace parts are described in detail in the text.

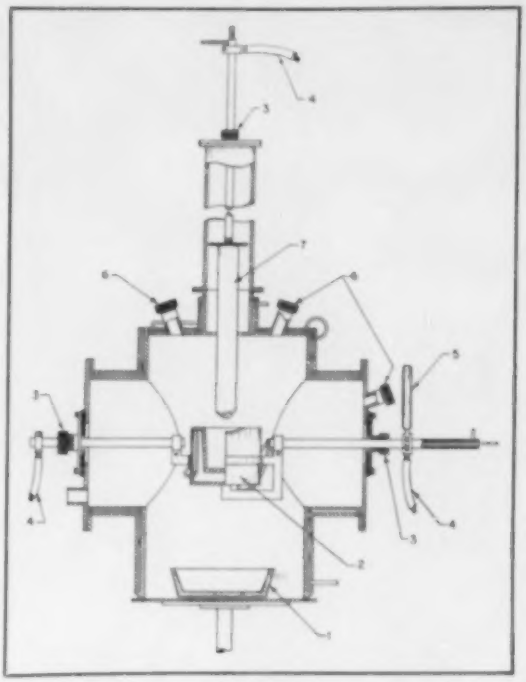

Fig. 6—Tilting crucible casting furnace consists of the following: 1. graphite mold, 2. tilting crucible, 3. O ring seal, 4. power connection, 5. tilting mechanism, 6. inspection port, and 7. consumable electrode.

Equipment

Two types of furnaces were investigated, one contained a bottom pour crucible, the other was a pour-over-lip type.

The bottom pour furnace, shown in Fig. 5, consisted of the following: A water cooled electrode holder (1) is suspended from a hoist and extends into the furnace through an O ring seal to support the consumable electrode. The cathode power lead (2) is connected to the electrode holder. Melting progress is observed through the sight glass (3) and the window (4). The furnace chamber and the mold box are evacuated through the ports (5) and

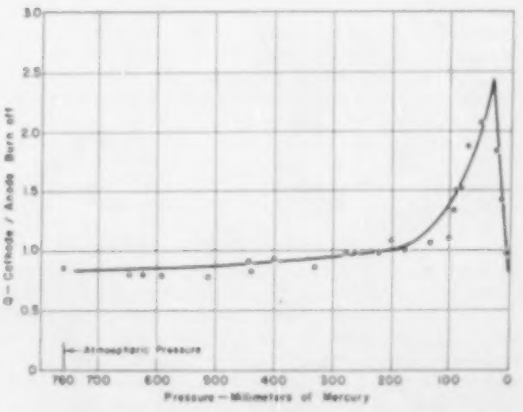

Fig. 7—Effect of pressure on melting rate shows that melting rate is enhanced by low pressure furnace atmosphere.

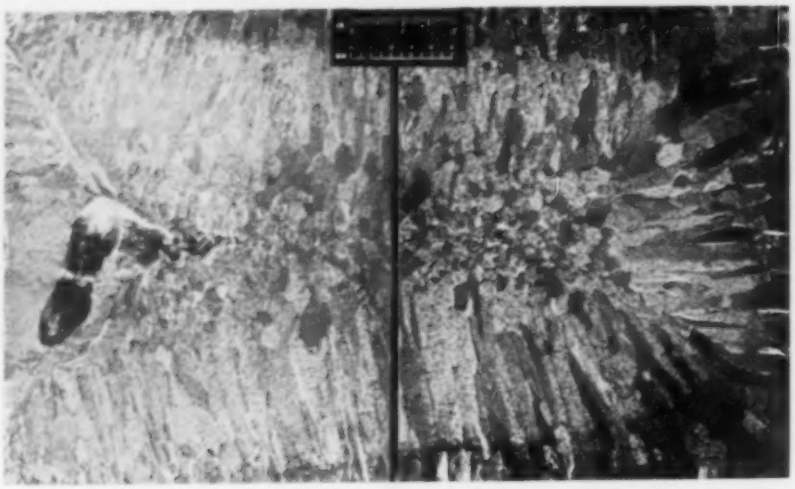

Fig. 8—Sectional ingot shows molten pool boundaries. This 10-in. diam melt had a pool depth of about 15 in.

(13). The water cooled crucible (7) serves as the anode and is insulated from the upper part of the furnace by an insulating ring (9). The anode power connection is at (6). A molten pool (8) results from melting the cathode and is retained by a water cooled stopper which can be manipulated by the outside controlled rod (11). The mold box (10) is held in place by a jack (14) and contains a mold (12) centered directly under the pouring gate.

In operation, a cathode large enough to provide a casting of predetermined size is suspended from the electrode holder, a mold is placed in the mold box, the furnace is sealed and evacuated to about 100 microns. After backfill with helium or argon, the arc is struck, and the desired amount of metal is melted into the crucible. The water cooled stopper plug is lowered and swung to one side and within a few seconds the skull melts through and the crucible drains into the mold.

The most serious problem in this type furnace is complete removal of drips from the tap hole which prevent the stopper from being reseated properly. Once this is overcome the ability to make several castings consecutively without opening the mold box will be a reality.

The pour-over-lip furnace contains a water cooled crucible that may be tipped by an outside controlled mechanism to pour directly into a mold placed inside the furnace. Details of the furnace are shown in Fig. 6. After an adequate amount of metal has been melted the electrode is raised and the casting is done at once. This pilot scale furnace is capable of producing 15 lb castings at a continuous rate of about 1 per hr. Currently a new furnace is being made that will have a ladle capacity of about 165 lb, of which 65 to 90 lb will be molten and available for casting.

Deep Pool Studies

Experimental results show that the maximum melting rate yields the greatest pool depth, and this is attained by melting in a low pressure atmosphere only slightly above the pressure where a glow discharge develops. This is in the neighborhood of 30 mm Hg, as indicated in Fig. 7. This then is the crux of the process. Consumable electrode melting alone could not provide enough heat to yield a deep superheated pool of molten titanium. However, by use of a low pressure furnace atmosphere, the melting rate is enhanced, and the furnace efficiency increases to the point where casting becomes feasible. Beall, Borg, and Wood[8] report melting efficiencies over 50 pct, yielding molten pools 8 in. deep in an 8-in. diam crucible, using the low pressure technique. Later studies indicated that it was possible to achieve a 15-in. deep pool in a 10-in. diam crucible, the molten metal amounting to about 63 pct of the ingot.

Pool depths were measured by melting under controlled conditions and dropping a piece of metal, usually manganese or copper, into the melt just as the arc was extinguished. The resulting ingots were sectioned and etched to reveal the pattern shown in Fig. 8. This 10-in. diam melt had a pool depth of about 15 in. It is estimated that 63 pct of the charge, or 120 lb of titanium, was molten at one time.

Melting rates of as much as 9 lb per min were recorded when 9000 amp at 30 v was used. This high power input provided enough superheat to insure good fluidity in the pour.

Acknowledgments

Appreciation is extended to Watertown Arsenal, sponsors of this project, for permission to publish this paper, and to S. V. Arnold under whose advisement the work was done. To H. L. Gilbert and J. O. Borg, formerly with the U. S. Bureau of Mines, and to others on the staff at Albany appreciation is also extended.

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Battery of crucibles in tilting furnaces is shown in a nonferrous shop.

Carbon Bonded Silicon Carbide Crucibles Offer Outstanding Advantages

by G. J. Easter

WHEN the patents covering carbon bonded SiC refractories expired practically all of the American producers of clay graphite crucibles added this new refractory to their line. It soon became obvious that these crucibles were longer lived than any former type.

In the first place, the carbon bonded body has a phenomenally low coefficient of thermal expansion of 2.7×10^{-6} per °C which gives it outstanding resistance to thermal shock. This property is even more conspicuous here than in carbon electrodes, perhaps because the presence of graphite flakes in the crucible wall seems to result in a sort of mechanical flexibility which also plays its part in preventing cracking by heat shock. This resistance to heat shock was accidentally demonstrated in the case of furnace door brick of this material which was dropped in water from 2500°F, then dried out, and put back in service without harm.

Crucibles made of carbon bonded SiC refractories can be put directly into a hot furnace without preannealing or preheating. A charge of cold metal can

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Fig. 1—Originally crucibles of carbon bonded type were limited to sizes that could be handled with tongs or shanks.

also be charged into the hot crucible and full heat applied without hesitation. The high thermal conductivity of 100 Btu's makes melting under such conditions very rapid, and the infusibility of the body permits extreme furnace temperatures to be employed without harm.

The chemical nature of the material is also particularly well fitted for use in crucibles. There is nothing present which molten aluminum can reduce to damage the wall as it tends to do with siliceous

Table I. Chemical Analysis of Carbon Bonded SiC Refractories, Pct

Material	Range	Avg
SiC	41 to 53	45
C	27 to 34	30
SiO_2	10 to 14	12
Al_2O_3	2 to 12	6
Fe_2O_3	1 to 1½	1

bodies, and except against basic slags or molten iron, the crucible is almost entirely inert.

The rigidity of these crucibles even at temperatures of 2700°F is such that damage from distortion by the tongs used to handle them in and out of the furnace is practically unknown.

Carbon bonded crucibles are used for all the nonferrous metals but particularly for the copper alloys and for aluminum. The latter has become tied in with crucible size ratings. Up to roughly No. 600, the number assigned to the crucible size is roughly its capacity in pounds of molten aluminum. For other metals an appropriate multiplying factor must be used, for example about three for red brass. In the larger sizes, numbers are arbitrarily assigned but high numbers mean big pots.

The first use of really large pots (1000 lb aluminum capacity) came during World War II when foundry capacities were being pushed to their utmost. These were so successful from the start that their use spread rapidly and crucibles with a capacity of from 1000 to 1500 lb of aluminum are now widely used for both aluminum and the copper group of metals. One plant uses three such crucibles simultaneously to pour bronze castings weighing 8000 lb each.

Advantages of Crucibles

Reliable recent figures on the cost of melting metal in crucibles vs the cost in electric furnaces are not available, but it seems to be generally conceded that crucibles offer outstanding advantages under the following conditions:

Fig. 2—Carbon bonded crucibles are heated in a pit or pot furnace usually with oil or gas flame.

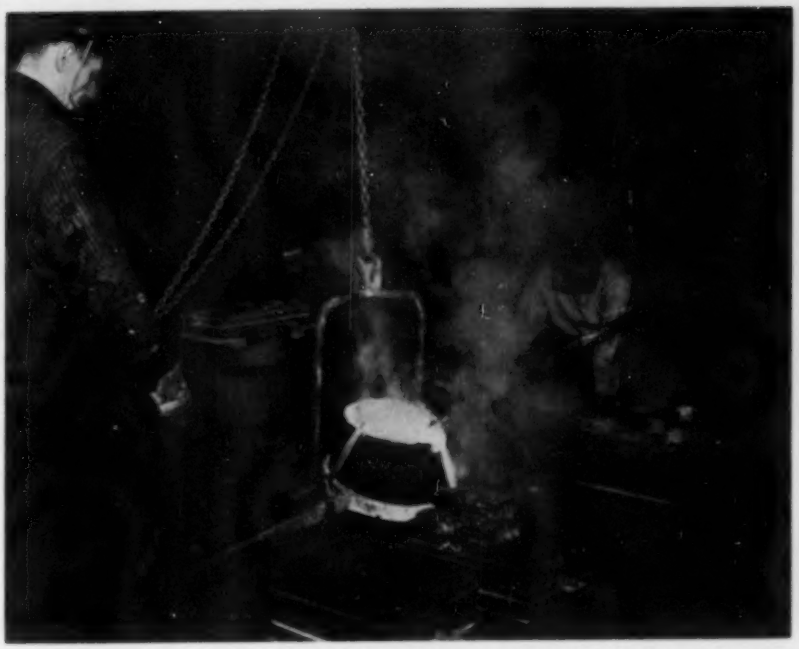

Fig. 3—When metal is ready to pour the crucible is transferred to a shank and transported by overhead monorail to the point of pouring.

A—Where relatively short runs are made before metal composition is changed. Usually the former metal is simply cleaned out and the same crucible used for the new type. Where ultra purity is required, a new crucible can be substituted without very high cost.

B—Where metal requirements are up and down in the course of business, particularly where only single shift operation is warranted. The overhead on electric induction furnaces and usually the electric demand charge go on regardless of business volume and put a serious penalty on intermittent operation. With crucibles there is no demand charge and overhead is a small fraction of that for electric equipment.

C—Where there is an urgent unforeseen immediate demand for more metal. Crucible furnaces are readily available from several suppliers or can be built much faster than either stationary reverberatory furnaces or electric induction furnaces. Thus they are obtainable in minimum time at moderate cost so that production can be increased quickly. Moreover, with them the need for specially skilled labor is kept at a minimum.

D—In shops where the large capital investment required by electric furnaces is simply not available. It is estimated that an electric furnace to deliver

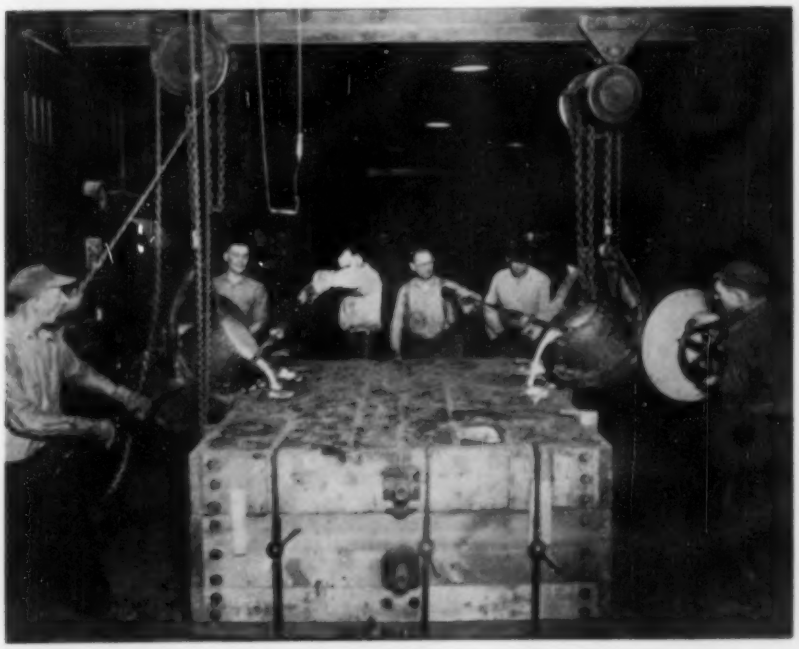

Fig. 4—Very large castings are sometimes made by simultaneously using two crucibles on opposite sides.

Fig. 5—A crucible in a tilting furnace is shown. Thorough cleaning and mechanical common sense extend the life of crucible.

1000 lb of brass an hr would cost from $13,000 to $15,000 whereas an equal capacity crucible set up would not exceed $2500.

Open flame barrel furnaces are also cheap initially but metal losses with them often reach 3 pct vs ½ to 1 pct for crucibles, and this difference quickly becomes unbearably costly. Crucibles are not always the final answer but they have proved to be a good answer in hundreds of shops, large and small.

Manufacture of Crucibles

Refractories of this type are made by dry mixing the granular ingredients, then mixing in premelted pitch or tar and allowing the mixture to age for several hours. Tables I and II give the chemical and physical properties of SiC refractories. While it is still warm enough to be plastic under pressure, it is formed to shape. Crucibles are formed in a barrel-like mold which rotates about its axis much like a potter's wheel while the mix is pressed against the side. In some instances, a separate lining is formed within the crucible at this stage using more basic materials such as magnesia or alumina (still with a carbon bond) to replace the acidic SiC. Such basic linings are used for melts high in nickel, chrome, or iron.

Until recently the size and shape of such crucibles were limited to those which could be formed in this way by rotary motion or which could be pressed. Improved techniques now permit molding irregular shapes, such as oval crucibles. Refractory blocks, runner troughs for copper and the like can be formed by any of the methods used in making plastic clay refractories although care has to be taken to secure complete internal cohesion. Overlapping folds tend to remain separate unless forcibly united.

Fig. 6—A crucible with lip cemented in place for use in a tilting furnace shows small base, a feature that was patented in 1951.

Table II. Physical Properties of Carbon Bonded SiC Refractories

Type	Data
PCE	>40
Porosity	27 to 30 pct
Weight	1.89 gr per cu cm 118 lb per cu ft
Specific heat to 2000°F	0.20
Coef of thermal expansion	2.7×10^{-6} per °F 1.5×10^{-6} per °F
Thermal conductivity	100 Btu /ft²/in./°F/hr 0.0345 cal/cm²/°C/sec
Modulus of rupture at 1350°C (2462°F)	470 psi
Cold crushing strength	1500 psi
Temp failure under 25 psi load	>3000°F
Wt of 9-in. straight	6¾ lb
Electrical resistivity	0.012 ohm/cm² cold 0.0076 ohm/cm² at 2200°F
Temp range of use	1300 to 3200°F (special glazes go lower)
Cold permeability to air	75 cu cm per min through 9x4½ in. block 2½ in. thick at 10 cm water pressure
Resistance to slag	Attacked by basic slags. Inactive to others

The molded pieces are baked under reducing conditions slowly enough to permit the volatile materials in the bond to be driven off without damage to the structure of the refractory. Heating is continued for at least 200°F above the temperature at which the last vapors come off. This yields a completely carbonized binder in a body containing 27 to 30 pct open pores.

The carbonized body is cooled and glaze applied, after which it is refired to a temperature which will mature the glaze. After that it is finally cooled down, inspected and shipped.

Crucible Use

Originally crucibles of the carbon bonded type were limited to sizes which could be pulled from the hot furnace and manually handled with tongs or shanks while the metal was being cast as shown in Fig. 1.

In this sort of application, the crucible is heated in a so-called pit or pot furnace, illustrated in Fig. 2, usually with oil or gas flame which swirls around it and finally comes out through a hole in the center of the furnace cover. The metal may be covered at this stage with molten window glass or other flux to assist in keeping down the oxidation losses. Most

fluxes are so basic in nature as to attack the crucible so that crucible life is shortened and most users prefer not to use flux. When the melt is ready to pour, the crucible is lifted out of the furnace with tongs. It is transferred to a shank, shown in Fig. 3, so as to permit easy transportation on an overhead monorail to the point of pouring where two men with the crucible between them can readily tip it. Any cover of flux or oxidized metal is first skimmed off using a skimmer made of the same material as the crucible, and castings are then poured. Very large castings are sometimes made by pouring simultaneously from two crucibles on opposite sides. This is illustrated in Fig. 4.

When it is desired to impregnate the molten metal with a highly oxidizable ingredient just before pouring, a charge of that ingredient is stuffed into an inverted cup of the SiC refractory. This is attached to a steel or graphite rod and the desired ingredient held below the surface of the metal in the crucible until it dissolves. Such a cup is called a phosphorizer but is used to add many things beside phosphorus, for instance, calcium boride which is an excellent deoxidizer for copper intended for electrical uses.

Before the empty crucible is returned to the furnace for another charge, it should be completely emptied and any adherent slag or dross removed. A very small expenditure of time on such cleaning pays large dividends in increased crucible life. It is particularly important, in fact essential, that the crucible be cleaned out thoroughly before being set aside for the day and allowed to cool down. Otherwise the thermal expansion of the adherent oxides and metal in the pot (which is several times that of the material of the crucible) is very likely to split the pot or break off pieces of it long before its potential life has been realized. A heel of metal which has been allowed to freeze in the pot is particularly likely to split the pot on reheating. The necessity for regular and complete cleaning out of the adhering nonmetal deposits is of course most urgent in aluminum work where the metal oxidizes to a very hard and refractory oxide which can never be removed except by scraping.

Fig. 7—Special crucibles called bowls are provided die casters to hold molten metal for long periods at low temperatures.

Another malpractice which ruins more crucibles prematurely than any other is the careless charging of ingots to be melted. It would seem obvious that they should not be dropped or thrown carelessly in such a way as to strike the crucible a hammer blow or tilted in such a position that they become wedged crosswise in the pot where their high thermal expansion may shove a hole right through the crucible wall. Many instances are reported each year where unduly short life of a crucible is due to exactly these causes.

The necessity for thorough cleaning and mechanical common sense in handling extends also to cru-

Fig. 8—Crucible for special uses shows passage leading down the wall to the bottom of the melting chamber. Molten material can thus be poured with minimum exposure to air.

Fig. 9—Special crucible made for aluminum industry permits smooth, rapid pouring with great accuracy.

cibles in tilting furnaces. One containing a crucible is shown in Fig. 5.

A crucible with a lip cemented in place for use in a tilting furnace is shown in Fig. 6. Attention is called to the small base on this crucible, a feature patented in 1951. This permits more space for combustion of the oil or gas flame around the base of the crucible and presents maximum area of contact of the flame with the lower part of the crucible rather than shielding the latter on a large pedestal block underneath a big flat bottom. Experience has shown that not only is fuel consumption lessened and combustion improved with these small bottom pots, but the life of the crucible itself in terms of metal melted is greatly prolonged and metal melting rate per hour is increased.

Crucible Life

Crucible life depends upon so many individual conditions that no generalization can be made. Melting point and reactivity of the metal involved, charging practice, fuel used, position of burners, etc. all enter into the answer. Crucible life in tilting furnaces is longer than in pull pots and is longer with aluminum than with brass. It becomes progressively less as increasing proportions of iron or nickel are in the melt. In the case of aluminum in tilting furnaces with gas or oil heat under ideal conditions there are numerous records of crucibles holding 1000 lb of metal that lasted for over 500 heats.

In tilting furnaces, it is common to space the crucible from the furnace wall by two refractory pillow blocks protruding from the wall against which the crucible rests when tilted. It should, however, not be wedged tightly from behind. Care should also be taken that the cover is not clamped tightly against the crucible which might crack. Where extreme protection from oxidation of the melt is required, a separate light lid made for the purpose may be rested directly on the pot. This should be the same material as the crucible.

Another precaution to be taken in using crucibles relates to the pedestal block on which the crucible is supported. This should be made of the same material as the crucible, as pedestals with higher thermal expansion are apt to pull the crucible apart upon heating. The pedestal should be flat and of sufficient area to completely support the base of the pot. Cross grooves are sometimes provided though these are not considered necessary with small bottom pots. Some foundries make a practice of throwing a little charcoal or graphite on top of the pedestal each time before placing a crucible on it to minimize sticking. In tilting furnaces similar sprinkling is done or a heavy cardboard circle may be placed under the pot when first installed.

Direct flame impingement against the side of a crucible is apt to build up a heavy, tightly adherent deposit of carbon at the point of impact. This insulates the part of the crucible to which it adheres, so that when surrounding areas are highly heated serious thermal strains may be set up and the pot may be broken. At best the carbon interferes with the efficient use of the fuel and should be carefully avoided. In all crucible furnaces, there should be adequate combustion space all around the crucible and the furnace wall should be kept smooth and even to avoid local gas swirls and hot spots.

Oxidation

Carbon bonded crucibles, when properly used, are very resistant to oxidation. They do have a danger zone, however, in the temperature range of 1000°F to about 1400°F where the tendency of unprotected carbon to oxidize is material and where the glazes usually used are not fluid enough to soften and give protection. Most crucible makers supply special glazes on pots intended for use in this range if they are apprised of the condition to be met.

For this same reason it is best to *sock* the heat to the metal to be melted rapidly rather than to *baby* the crucible in an attempt to prolong its life. Less harm is done by the sudden temperature change than by extended holding in the oxidation zone. Holding molten metal for a considerable time is bad for both the metal and the pot if the atmosphere is highly oxidizing, so when fuel is shut down, the air should be cut back proportionately. Flame size and nature should be so controlled as to maintain the furnace atmosphere barely on the oxidizing side.

Special crucibles shown in Fig. 7 called bowls are provided die casters and others who desire to hold molten metal for long periods at low temperature. When properly glazed the life of such a bowl should exceed that of the crucible in which the metal is melted down. Resistance type electric furnaces are sometimes used to heat bowls of this type. Oxidizing conditions in these furnaces are especially severe and careful glazing with a suitable glaze becomes very important.

Refractory crucibles of the type discussed here have not in general been used in large high frequency electric furnaces for melting nonferrous metals. In such cases rammed cement linings fired directly in place are preferred. There are however numerous users of carbon bonded crucibles in small high frequency furnaces (up to crucible size 70). Furnaces using them have to be specially insulated to prevent high heat losses to the coil through the high thermal conductivity crucible as well as to keep the crucible from shorting the electric coils.

Special crucibles sometimes encountered are shown in Figs. 8 and 9. The one in Fig. 8 has a passage leading down the wall to the bottom of the melting chamber so that molten material may be poured with minimum exposure to the air while slag inclusions are kept floating well out of the working area. Fig. 9 shows a special crucible made for the aluminum industry which permits smooth, rapid pouring with unusual accuracy.

Effect of Certain Primary Mill Heating and Rolling Practices On Slab Surface Quality

by H. B. Wishart and C. A. Hope

THE number and severity of surface imperfections on rolled slabs, assuming the reception of uniformly good quality heats from the open hearths, depend upon a number of conditions associated with heating and rolling of steel in the primary mills. While there are other elements that have an effect on surface quality, rather than make this paper too lengthy the discussion of mill processing variables will be limited to three factors in order to illustrate their effect on slab surface quality: 1—amount of reduction between ingot and slab, 2—slab finishing temperatures, and 3—influence of roll changes.

Measurement of the percentage of surface removed by scarfing on the flat surfaces of the slabs was used for evaluating quality in this study. A study of this nature must be made on slabs that are inspected and conditioned under uniform conditions and standards. In order to be certain that steel of the best possible steelmaking quality was received at the rolling mills, a low carbon deep drawing grade of rimmed steel, made for sheet mill application and manufactured under carefully controlled open hearth practices, was selected. Use of this particular steel thereby reduced the steel quality factor as a possible source of variation.

The measure of slab surface quality is the amount of surface removed by scarfing during yard conditioning and is expressed as percentage of surface removal. This percentage was estimated by experienced personnel to uniform standards and it should be noted that these percentage figures are representative only for the grade selected and under the rigid inspection standards to which the slabs were subjected. Surface removal is a matter of concern because the imperfections, if undetected, often are the cause of poor surface quality in the finished product and their removal, by conditioning, add to the total plant costs.

The data was collected at various times over a period of six months and, with normal weekly fluctuation in the level of quality, it is to be expected that for any period of sampling there will be some variation in the plotted results.

The type of defect that is responsible for most slab conditioning at Gary Steel Works is a light angular surface break. This defect is usually interconnected to form large defective areas and occurs mainly on the bottom half of the slab on both sides. This type of defect is illustrated in Fig. 1, showing a bottom portion of a slab; while a close-up of an area of the same slab where the surface breaks were found is given in Fig. 2. The large dark areas are heat patterns, results of hot stacking and cooling.

Independent of these light breaks, but sometimes associated with them, are larger and deeper isolated breaks. However, most of the surface removal occurs from conditioning the large areas of small breaks.

There are also occasional scabs of varying degrees of severity but these are relatively infrequent. The slabs are generally free of rolled-in scale.

Ingot to Slab Reduction vs Quality

Some ten years ago at Gary Steel Works, the minimum slab ingot thickness was 30 in. Subsequent operating and other quality features indicated the desirability of using an ingot of reduced thickness with the result that a series of molds that would produce a 22 in. thick ingot with various widths was adopted as standard. A minimum number of the thicker mold sets are still used. From the standpoint of quality difficulties from pipe and segregation, the smaller ingots are an improvement. From a quality surface viewpoint, the slab product from the thin molds requires approximately twice the surface removal as the slab product from the thick molds. This is shown in Fig. 3 where the percentage of surface removal is plotted against slab product rolled from

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Fig. 1—View of the bottom portion of a slab shows defective area.

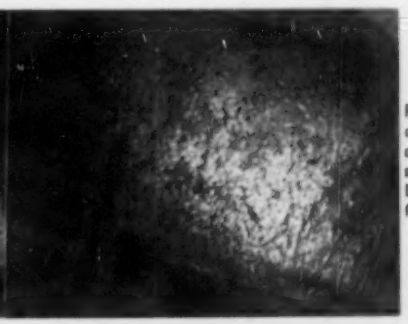

Fig. 2—Close-up of the same slab as in Fig. 1 shows surface breaks in the defective area.

The Transactions papers appearing in the Journal of Metals during 1955 will subsequently appear in AIME Transactions Volume 203, and may be permanently referenced to that volume.

various size ingots ranging from 22 to 35 in. thick. In order to obtain a representative number of samples from slabs rolled from different size ingots for the data contained in Fig. 3, various grades of rimmed steel were used for this illustration.

That less reduction has a detrimental effect on surface quality can be further seen from Fig. 4 where surface removal is shown to increase as slab thickness increases (or slab reduction diminishes). The figure also illustrates surface quality in relation to slab width. The slabs in Fig. 4 were all rolled from 22 in. thick ingots.

The amount of side reduction in rolling to finished width has little association with defects on the flat slab surfaces. However, slab rejections, caused by massive edge breaks, are influenced considerably by the amount of side reduction. The less the side reduction from ingot to slab, the higher will be the rejections.

The authors do not wish to convey the impression that good surface quality cannot be obtained from sheet, strip, or plates rolled from slabs produced in the 22 in. series ingots, because it can. Rather, the data indicate that the amount of slab conditioning necessary to provide a good rollable slab is a direct function of amount of reduction from ingot to slab.

Slab Finishing Temperatures vs Quality

With the installation of automatic recording pyrometers on the exit side of the mills, it was found that slabs finished at a high temperature had proportionately better surface quality than slabs finished at a low temperature. This is illustrated in Fig. 5 by 3260 slabs that had a finishing temperature ranging between 1900° and 2150°F.

Higher soaking pit control settings are an aid in securing better finishing temperatures but, since the increase in possible pit temperatures is limited, such settings, by themselves, cannot be considered the only means for securing higher finishing temperatures. Other operating factors that have an effect on the temperature are proper scheduling of heats through the pits, general operating efficiency of the pit crews, bunching of ingots on the entry side of the mill, slow delivery from pits to the mill, improper spacing of the ingots in the pits, and maintaining waste-gas ports and recuperator tubes at maximum operating efficiency.

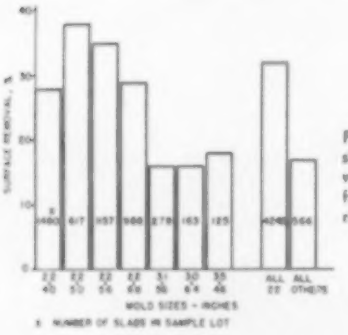

Fig. 3—Percentage of slab surface removal vs ingot size is given for various grades of rimmed steel.

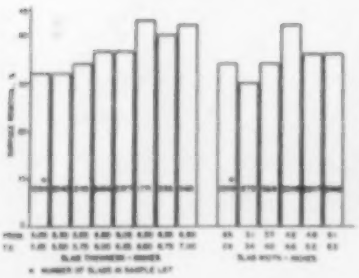

Fig. 4—Slab surface removal vs slab thickness and width is given for 0.05 to 0.08 pct C drawing quality rimmed steel. All slabs were rolled from 22 in. thick ingots.

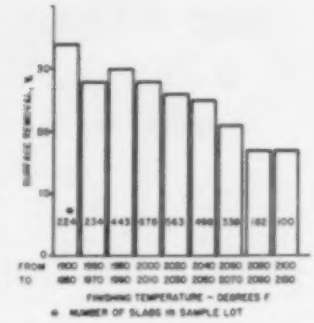

Fig. 5—Slab surface removal vs slab finishing temperatures is given for 0.05 to 0.08 pct C drawing quality rimmed steel. All slabs were rolled from 22 in. thick ingots.

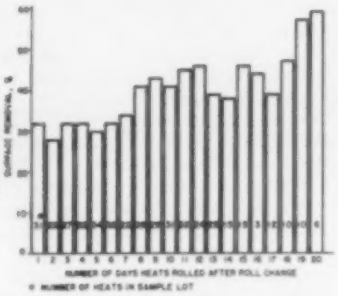

Fig. 6—Slab surface removal vs roll changes is given for 0.05 to 0.08 pct C drawing quality rimmed steel. All heats were poured in 22 in. molds.

Rolling slabs to the high side of the finishing temperatures will require maintenance of the pits at a higher level of operating effectiveness but the results should be reflected in lower operating costs and improved quality.

Influence of Roll Changes vs Quality

The length of time that a set of rolls remains in use is another factor that influences slab quality. The advantages of maintaining good roll surface by changing the sets at more frequent intervals is illustrated by data secured from 438 heats finished on the 36 in. mill.

The surface removal of each heat was plotted against the number of days that the set of rolls had been in use, as shown in Fig. 6. From the data plotted in this figure, there is an indication that, as a set of rolls wears between periodic dressings, the surface quality of the slabs is adversely affected if the rolls are used for an extended period of time.

Summary

The data presented show that surface quality of slabs is affected by the amount of ingot to slab reduction, slab finishing temperature at the last pass on the mill, and by the frequency of roll changes. As previously mentioned, there are other factors in the course of making and rolling a heat of steel that can have a detrimental influence on slab surface quality. The three selected for presentation in this paper are only a part and, in the last analysis, the optimum steel surface can only result when proper control is exercised over all the factors known to affect the quality of the grade under consideration.

Discussion of this paper, if any, will appear in JOURNAL OF METALS, November 1956, and in AIME Metals Branch *Transactions*, Vol. **206**, 1956.

Aerial view of the Flin Flon electrolytic zinc plant of the Hudson Bay Mining and Smelting Co. Ltd. shows most of the metallurgical process units. From left to right are shown the fuming plant and baghouse, copper smelter, and sulphide and oxide leaching plants. Stack seen in foreground is that of the copper smelter and the one in the right background that of the zinc sulphide roasters.

Processing of Zinc Oxide Fume at Flin Flon, Manitoba

by Roy Ellerman

The hydrometallurgical processing of an impure zinc oxide fume is described. Flowsheet includes roasting for fluorine elimination and countercurrent leaching to produce a neutral sulphate solution. This solution is combined with zinc sulphide leach solution for subsequent purification and electrolysis. Novel to the neutral leach step is the use of automatic pH control.

THE electrolytic zinc plant of the Hudson Bay Mining and Smelting Co. Ltd. was erected in 1929–1930 to process zinc sulphide concentrates produced from ore mined at Flin Flon, Manitoba. Since 1930, production of high purity slab zinc has been continuous. In May 1951, construction of a fuming plant was completed and operations were started to recover zinc from copper smelter reverberatory slag. A second zinc-bearing material in the form of an impure zinc oxide fume was then made available for treatment. Coincident with the construction of the fuming plant, additions were made to the electrolytic zinc plant to recover the zinc in the fume as slab metal.

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Discussion of this paper, TP 4034D, may be sent, 2 copies, to AIME by Sept. 1, 1955. Manuscript, Nov. 29, 1954. Chicago Meeting, February 1955.

The original plant capacity of 70 tons of slab zinc per day was steadily increased up to May 1951, at which time the daily production averaged 130 tons. With the advent of the fuming plant and the processing of zinc oxide fume, production was stepped up to the present capacity of 190 tons per day. Of this, approximately 48 pct is contributed by the processing of zinc oxide fume. This additional source of raw material has allowed a 20 pct cutback in mining and milling operations, thereby adding considerably to the life of the Flin Flon mine.

Zinc sulphide concentrates are treated by the usual hydrometallurgical method. Details of this operation and plant were described by Carr and Reikie in 1935[1] and Kent in 1951.[2]

The impure fume is treated in two operations. First, it is roasted to eliminate fluorine. Next, it is subjected to a countercurrent leach with spent electrolyte to extract the zinc as a neutral zinc sulphate solution. During this leaching step, soluble impur-

ities such as arsenic, antimony, and germanium are precipitated. The degree of purity enables this solution to be combined directly with the sulphide leaching plant solutions. The combined flow is then purified and electrolyzed to produce high purity zinc. Up to the present time, no build-up, or increase, in the level of impurities in the finished electrolyte has resulted from using the *combined flow* method of treatment. Fluorine control has never been a serious problem at Flin Flon.

Roasting Process

Flowsheet of the zinc oxide roasters is given in Fig. 1. The primary purpose of the roasting operation is to reduce the fluorine content in the fume to a level where it does not interfere with the electrolysis of the zinc sulphate electrolyte produced by leaching this fume. Experience has shown this level to be 0.003 pct. During electrolysis, minor quantities of fluorine in zinc cell electrolyte accelerates corrosion of the aluminum cathodes. This corrosion is generally accompanied by sticking of the zinc deposit to the aluminum sheet. In Flin Flon, difficult stripping has been experienced with a fluorine concentration of 20 mg per liter in the cell electrolyte. Elimination of this impurity from the fume prior to leaching is essential, since there is no satisfactory method of removing it from solution to the level required. Roasting of the fume has consistently given 99.9 pct fluorine elimination.

The source or sources of fluorine in the fume are difficult to trace due to assay limitations. Copper concentrate, with its burden of insolubles, silica flux and zinc residue, that go to make up the copper smelter charge as well as the coal used for smelting, probably all contain minor quantities of this element. Concentration of these stable fluorides will occur in the reverberatory slag that becomes the fuming furnace charge. Under extreme reducing conditions in the fuming furnace bath, fluorine is driven off in a gaseous form and concentrates by absorption on the zinc oxide particle. Since the fluorine can be driven readily from the fume by heat, it is assumed that it does not combine to form stable compounds with any of the constituents in the fume.

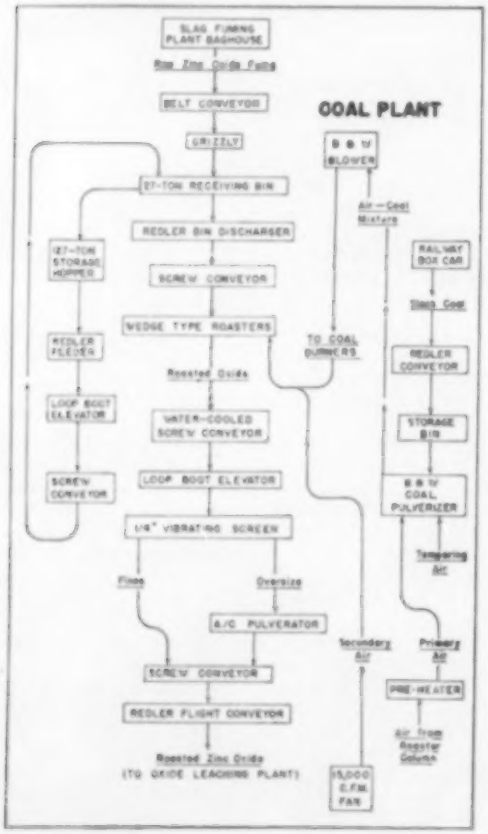

Fig. 1—Flowsheet of zinc oxide fume roasters illustrates the process for fluorine elimination.

An average of 115 tons per day of impure zinc oxide fume is received from the fuming plant baghouse. The assay of this material appears in Table I. Continuous delivery of the fume is made to the top of the roaster building by a 24 in. belt conveyer over a distance of 1305 ft. From the belt, the fume drops through a grizzly into a 27 ton capacity receiving tank. This tank is equipped with a double outlet "Two Stage" type Redler bin discharger and arch breaker operated by a manually controlled variable speed dc motor. Variable rates of feed up to 10 tons of raw fume per hr can be obtained with this type of feeder mechanism. Excess fume from the tank, over and above that fed to the roasters, may be bypassed by gravity to a 120 ton storage bin. This rectangular hopper-bottomed bin is equipped with a 9 in. horizontal-type Redler feeder that discharges the oxide to a 9 in. *loop-boot* elevator. Rated capacity of both feeder and elevator is 10 tons of raw fume per hr. The discharge from the elevator is returned to the receiving tank by screw conveyer. The storage bin was deemed necessary to store fume in the event that one or both roasting units were unserviceable for short periods of time.

From the receiving tank, a regulated amount of fume is discharged to the feed hearths of two Wedge-type roasters by short 12 in. screw conveyers. Manually operated metal slide gates on the outlet of the bin discharger are used to split the feed to each roasting unit.

Table I. Average Assay (for 1953) of Zinc Oxide Fume Before and After Roasting

Material	Pct								
	Zn	Pb	Fe	Cu	As	Sb	Ge	SiO_2	F
Zinc oxide roaster feed	71.1	2.2	0.9	0.16	1.1	0.34	0.003	1.4	0.18
Roasted zinc oxide	73.8	2.0	1.1	0.18	1.0	0.30	0.002	1.7	0.002

The physical nature of the raw fume makes mechanical handling very difficult because of its tendency to pack into a lightweight integrated mass. The flow characteristics of the material are poor and it arches badly in bins. These difficulties are enhanced by long periods of storage. It has become a plant operating policy to keep the fume moving at all times and to use storage facilities only when necessary. Little difficulty has been experienced provided there is a continuous movement of the material and continuous operation of the arch breaker.

The two modified Wedge-type roasters are 25 ft diam, 31 ft high, and have seven burning hearths and a top feed hearth. Total hearth area of each unit is 2533 sq ft. The flue offtake is from No. 1

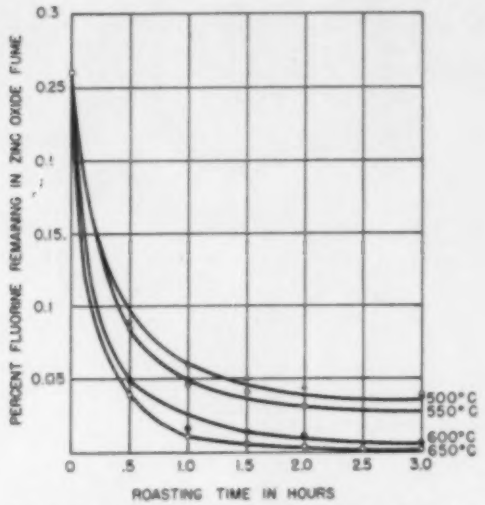

Fig. 2—Curves show the elimination of fluorine from zinc oxide fume in the laboratory kiln as a function of time at four temperatures.

hearth and discharges directly into the main zinc sulphide concentrate roaster flue. The insulated center column, to which are attached air-cooled rabble arms, revolves at one revolution per 45 sec. The fourth and seventh hearth each have four rabble arms, the others two. Plugging of the charge between the rabble blades is evident only on the fourth hearth due to high temperatures. The use of four rabble arms with rabble blades alternated between spacers has minimized this problem. The use of four rabble arms on the seventh or bottom hearth ensures a more even rate of discharge. Rabble arms and roaster column lute rings are of cast alloy iron containing 1 pct Cr, 1 pct Ni, 2.5 pct Si, 0.6 pct Mn, and 3.5 pct C. Serviceable life of the arms under continuous operation is about three years; lute ring life is somewhat longer. Rabble blades give up to five years service and are cast from an alloy iron containing 28 pct Cr and 1.75 pct C.

Heat to the oxide roasters is supplied by a Babcock and Wilcox pulverized coal system. Coal consumption averages 147 lb per ton of oxide roasted. External Dutch oven combustion chambers are used to house the burners. Three of these chambers are evenly spaced around the outer roaster shell of both the fifth and seventh hearths. The system is equipped for either automatic or manual control, the latter generally being used.

Alberta slack coal is delivered from box cars to a 50 ton storage tank by an 11 in. flight conveyer. This tank has a built-in electronic control and feeder that supplies a regulated amount of coal to a Babcock and Wilcox type-B coal pulverizer. The powdered coal is air-conveyed through a diminishing diameter steel pipe that loops back to the pulverizer through a Babcock and Wilcox No. 42 coal-air blower. Offtakes from this main header supply a coal-air mixture to each of the Dutch oven burners. Warmed primary make-up air is drawn into the pulverizer from the center column of the oxide roasters. Available heat can be supplemented in cold weather by passing the primary air through a steam-operated preheater. Air preheating is necessary to maintain the operating temperature in the pulverizer at 160°F to prevent moisture condensation. Secondary air to the burners is supplied directly from fans at 3 oz pressure.

Two types of Dutch ovens are in use at the present time. One roaster is equipped with ovens constructed to direct the burner flame along a radius of the roaster cross-section. This type has a removable arched roof for quick cleaning. The second roaster is equipped with ovens constructed to direct the burner flame at a tangent to the roaster wall. This type of firing causes the hot gases to move in a circular path within the hearth space to give good heat distribution. Both types have construction and operating points in their favor.

Recently, plant-scale tests were conducted using fuel oil firing. The encouraging results obtained during this test have prompted conversion of both roasting units to this type of firing. Revisions are being made at the present time to burn Bunker C fuel oil in both oxide roasters.

Each of the two roasters is equipped with automatic recording temperature indicators. These record the temperature of hearths two to seven in sequence at 1 min intervals.

The hot roasted zinc oxide is discharged through a chute at the outer edge of the seventh roaster hearth to a 12 in. water-cooled screw conveyer. From the conveyer, a *loop-boot* elevator discharges the oxide to a ¼ in. Hum-mer vibrating screen. Screen oversize, amounting to 2 pct of the roaster charge, drops by gravity to an Allis-Chalmers No. 1 Pulverator Hammer Mill. Screen fines and hammer mill discharge drop to a common screw conveyer that transports the material to a 12 in. *flight* type Redler conveyer. The latter delivers the entire roasted product to the top of the oxide leaching plant building, a distance of 230 ft. The first 65 ft portion of this flight conveyer has water-cooled troughs. Conveyers, elevators, and vibrating screen mentioned here have a rated capacity of 10 tons of

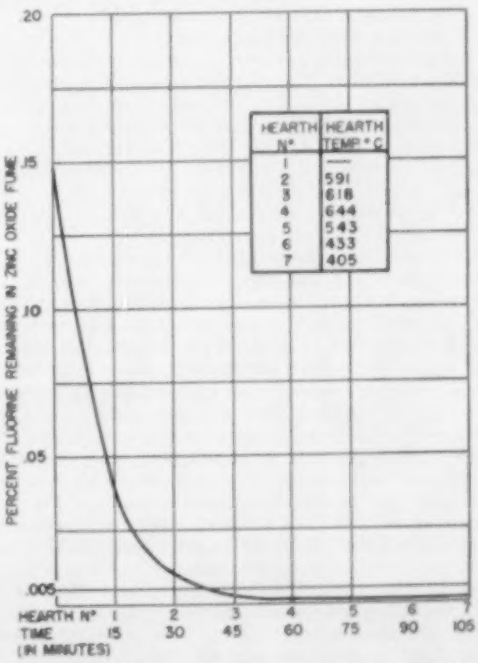

Fig. 3—Curve illustrates fluorine elimination from zinc oxide fume in the roaster as a function of time and hearth number.

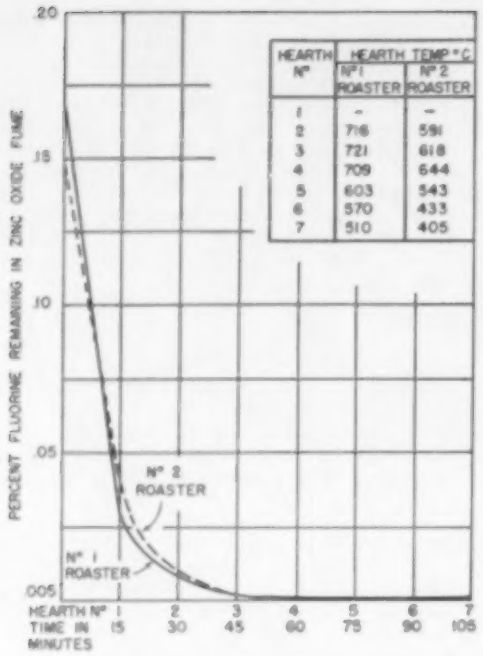
.20
.15
.10
.05
.005
PERCENT FLUORINE REMAINING IN ZINC OXIDE FUME
HEARTH N° 1 2 3 4 5 6 7
TIME IN MINUTES 15 30 45 60 75 90 105
N° 2 ROASTER
N° 1 ROASTER
HEARTH N° | HEARTH TEMP °C N° 1 ROASTER | N° 2 ROASTER
1 | – | –
2 | 716 | 591
3 | 721 | 618
4 | 709 | 644
5 | 603 | 543
6 | 570 | 433
7 | 510 | 405

Fig. 4—Curves show that increasing the roaster temperature above 650°C has little effect on the rate at which fluorine is eliminated from zinc oxide fume.

roasted oxide per hr. The hammer mill can handle well over 1 ton of lumped oxide per hr. This extra-large-sized mill can dissipate up to 12,000 Btu per hr, allowing the handling of a light load of hot material (500° to 600°C) without special cooling or materials of construction.

Roaster Operating Characteristics

During the first year of oxide roaster operation, experimental work was conducted, in both the laboratory and plant, to determine roaster operating characteristics. Investigations into fluorine removal as a function of time, temperature, and tonnage of fume treated were carried out. Attempts were also made to determine the temperature at which nodulizing occurs. The results of operating experience and test work have now made roasting of oxide a relatively trouble-free operation.

Tests for fluorine removal as a function of time and temperature were first conducted under close laboratory control, using a small rotary kiln. Samples of oxide were removed every 15 min and assayed for fluorine. Fig. 2 shows the effect of roasting time on fluorine elimination at various controlled temperatures. It can be seen that fluorine is virtually eliminated after roasting 2 hr at 650°C. This work was then transferred to plant operations where oxide samples were removed from the discharge of each hearth of one roaster and assayed for fluorine. The retention time for fume on any one hearth was calculated to be 15 min. The average roaster temperature for all hearths was 540°C with a high of 644° on the fourth and a low of 405° on the seventh. The results of these tests, Fig. 3, show that in actual plant operation fluorine removal is essentially complete after 1 hr, or after the oxide has been discharged from the fourth hearth. During this test, 57 tons of fume per day were treated in the roaster. Subsequent checks confirmed the foregoing results. Fig. 4 shows a comparison of roasting at two different average temperatures, each roaster treating essentially the same amount of fume. No. 1 roaster operated at an average temperature of 504°C with a high of 644° on the fourth hearth, while roaster No. 2 operated at an average of 638°C with a high of 709° on the fourth hearth. A study of this graph indicates that increasing roasting temperature above 650°C does not increase the rate of fluorine elimination or the amount removed.

The effect of increasing the tonnage treated in the roaster is to increase the depth of fume on the hearths without an appreciable change in retention time. Results of test work done to determine the amount of fume a roaster could handle efficiently is shown in Fig. 5. The tonnage of feed to No. 1 roaster was 90 tons and to No. 2, 45 tons during this 24 hr test. The curves show that increased roaster load or bed depth has the effect of shifting the location of optimum fluorine removal to a lower hearth, No. 5 in this instance. Increased temperature in No. 1 roaster does not appear to compensate for load increase. It can be assumed from the foregoing results that the oxide roasters could be operated efficiently with loads up to 90 tons per day with a margin of safety.

Attempts to determine the temperature at which agglomeration occurs in the roaster, in the size range above 1/16 in. mesh, did not meet with much success. Experience has indicated that temperatures above 750°C in the roaster result in a large increase in the amount of wet classifier sands in the oxide leaching plant. Therefore, high roasting temperatures are avoided. Agglomeration apparently results from the fusion of impurities in the fume, such as lead, whose oxides melt below 600°C. The particles formed by fusion contain about the same average zinc as the fine material. The percentage of fluorine

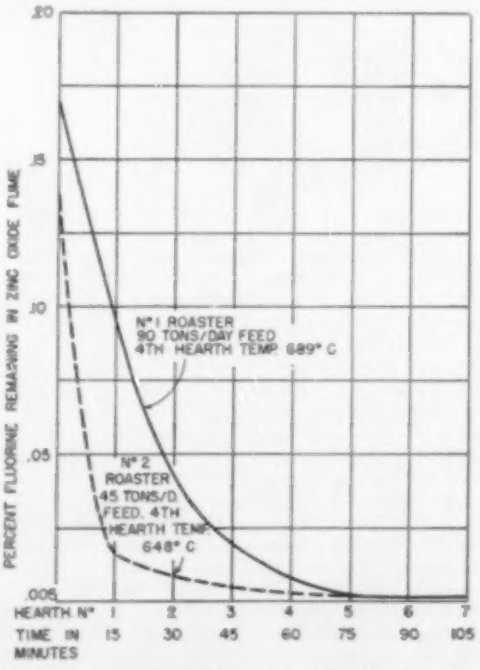
.20
.15
.10
.05
.005
PERCENT FLUORINE REMAINING IN ZINC OXIDE FUME
HEARTH N° 1 2 3 4 5 6 7
TIME IN MINUTES 15 30 45 60 75 90 105
N° 1 ROASTER 90 TONS/DAY FEED 4TH HEARTH TEMP 689° C
N° 2 ROASTER 45 TONS/D. FEED. 4TH HEARTH TEMP 648° C

Fig. 5—Curves illustrate the effect of fluorine elimination from zinc oxide fume in the roaster according to the tonnage of fume treated.

remaining in the lumps and fines, after roasting, is about the same, making retreatment of the hammer mill discharge unnecessary. The −¼ in. product that passes to the oxide leaching plant has not constituted a problem. It does not settle out in the leach tanks, can be separated from solution by a wet classifier, and is readily broken down with a weak sulphuric acid leach. A typical screen analysis of roaster products is given in Table II.

Test work has also indicated that, during roasting of the fume, apart from the elimination of fluorine, certain other benefits are derived. Impurities such as arsenic and antimony are rendered less soluble during leaching. The reducing power of the raw fume, due largely to absorbed SO_2 gas, is reduced to a trace, a reason why manganese dioxide is not required during the leaching process. The apparent specific gravity of the raw fume is increased from 30 to 70 lb per cu ft.

Dust burden of the flue gas leaving the oxide roasters has been found to average 1.7 pct of the tonnage of fume treated. Much of this drops out in the main flue system. Reabsorption of fluorine by dust collected from roaster flues and Cottrell precipitators is not evident.

Table II. Screen Analysis of Oxide Roaster Product

Screen Mesh, In.	Roaster Discharge, Pct	Vibrating Screen, Oversize, Pct	Vibrating Screen, Undersize, Pct
+1	0.1	4.8	0.0
+¾	0.2	11.8	0.0
+½	1.7	18.5	0.0
+¼	5.2	44.2	0.2
+⅛	7.6	10.0	3.0
+1/16	10.0	1.6	7.3
−1/16	75.5	9.1	89.5

The mechanical handling of light dry fume is accompanied by considerable dusting. To eliminate this condition as much as possible, all units such as the roaster feed hearths, conveyer transfer points, bins, etc., are covered and vented to a Draco dust-collecting system. This consists of a two compartment 24 bag unit. The dust is collected and returned to the roaster feed hearth by conveyer; the gas is discharged into the main roaster flue. Floors and equipment are kept relatively clean by a vacuum sweeper system having outlets at convenient points throughout the building.

Successful operation of the oxide fume roasters is a matter of maintaining proper roasting temperatures. Results of past experience have lead to certain fundamental rules of operation as applied to this plant:

1—To maintain a fluorine level in the finished electrolyte below 10 mg per liter and to prevent any build-up of this element in solution, roasted oxide fume must not contain more than 0.003 pct fluorine, and preferably 0.001 to 0.002 pct.

2—To maintain a consistently low fluorine content in the roasted product, No. 4 hearth should be maintained at a temperature in the vicinity of 650°C and No. 7 hearth should not be allowed to cool below 550°C.

3—Temperatures above 750°C should be avoided in order to prevent excessive agglomeration of the oxide in the roaster.

4—It is preferable to keep unroasted fume moving at all times in order to prevent packing and arching of the material in bins.

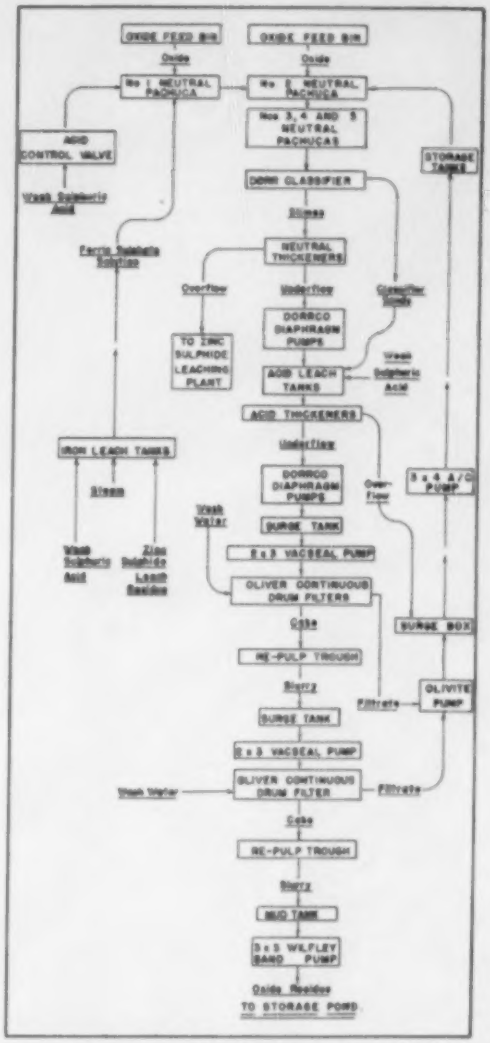

Fig. 6—Zinc oxide leaching flowsheet illustrates the two-stage countercurrent continuous process for extracting zinc from the roasted fume.

Zinc Oxide Leaching

Flowsheet of the zinc oxide leaching process is given in Fig. 6. The zinc oxide leaching process utilizes a continuous two-stage countercurrent leach for the extraction of zinc from the roasted fume. In the first or neutral stage, an excess of oxide is leached with return acid from the electrolytic cells to produce a neutral slurry. Thickening of this slurry gives a clarified zinc sulphate solution and an underflow containing undissolved fume and precipitated impurities. The neutral sulphate solution is laundered to the sulphide leaching circuit for subsequent purification and electrolysis. In the second, or acid stage, the neutral thickener underflow is leached in mechanically agitated tanks with a slight excess of cell acid to recover the bulk of the remaining zinc. The resultant slurry is thickened. Underflow from the acid thickeners is filtered and the filtrate, together with the acid thickener overflow, is returned to the neutral stage. The filter cake or *oxide residue* is stockpiled pending further treatment.

Neutral Leach Process: The roasted fume is delivered from the roaster building to the top of the oxide leaching plant by flight conveyer. Here it is discharged into two 25 ton capacity neutral Pachuca feed bins. Delivery surges may be diverted to any of three cylindrical steel storage tanks each having a capacity of 375 tons. These tanks also store the roasted fume during periods when the leaching plant is shut down, thus allowing the slag fuming plant and oxide roasters to continue to operate. Because of additional handling involved when the oxide is stored, the storage facilities are used only when necessary, and the material is not allowed to remain in them for any length of time. Each storage tank is equipped with a totally enclosed 7 in. horizontal Redler conveyer. This conveyer delivers the material to a 7 in. loop-boot elevator which in turn discharges the stored oxide to one or both of the Pachuca feed bins. All storage handling equipment has a rated capacity of 8.75 tons of oxide per hr.

The feed bins are located directly above Nos. 1 and 2 neutral Pachucas. They are equipped with bin dischargers and arch breakers operated by variable speed dc motors. From these bins, the roasted oxide drops to short type-316 stainless steel launders to be carried into No. 1 Pachuca by a flow of cell acid and into No. 2 by a combination of acid thickener underflow and residue filtrate solution.

Five Pachuca tanks, 10 ft diam by 30 ft stave, are connected in series by launders to form the neutral leach circuit. Based on a solution flow of 1100 water-tons per day, the total retention time of the pulp in the Pachucas is 4 hr.

Roasted zinc oxide fume contains very little soluble iron and it is necessary therefore to add this element to the Pachucas in the form of ferric sulphate solution. The amount used is 1 g of soluble iron per liter of Pachuca flow. This reagent is produced by leaching iron-bearing residue from the sulphide leaching plant. The leach is carried out at elevated temperatures in two mechanically agitated tanks using cell acid. The leach tanks are 12 ft diam by 12 ft, one being of steel construction having a homogeneous lead lining while the second is of wood stave construction lined with acid-proof brick backed by rubber. A measured amount of residue from the sulphide leaching plant is pumped as a heavy slurry to either of the two tanks. Cell acid is added and the temperature raised by steam injection to 70°C. Leaching is allowed to continue for approximately 12 hr, at which time the slurry is discharged at a controlled rate to the first neutral Pachuca. Tanks are charged and discharged alternately as required.

The pH of the Pachuca pulp appears to be the controlling factor as to whether the iron precipitation reaction proceeds. Plant practice of discharging the final neutral Pachuca at a pH of 5.2 to 5.4 has consistently produced good leaching results. Indications are that with a pH of less than 5.0 there is danger of a large increase of impurities such as arsenic, antimony, and germanium appearing in the neutral leach solution.

The neutral leach circuit is operated automatically by a two-step pH control system. Control points are located at the pulp discharge outlets of Nos. 1 and 2 Pachuca. Each step is independent of the other.

In the first step as shown schematically in Fig. 7, a temperature-compensated pH electrode assembly dips into the pulp discharging from No. 1 neutral Pachuca. The pH of the pulp is measured using a type "R" Beckman continuous pH indicator. The electrical impulse from the Beckman indicator is amplified, recorded, and converted to an air pressure in a Bailey controller. Air pressure variations, corresponding to pH variations, are relayed through a Bailey Standatrol unit to an air-operated Saunders-type valve located in the 6 in. acid line to this Pachuca. The Bailey selector valve allows the equipment to be operated manually if desired but this is only done when repairing or maintaining the electrode assembly. The Thy-mo-trol control unit, which controls the oxide feed rate, is independent of the

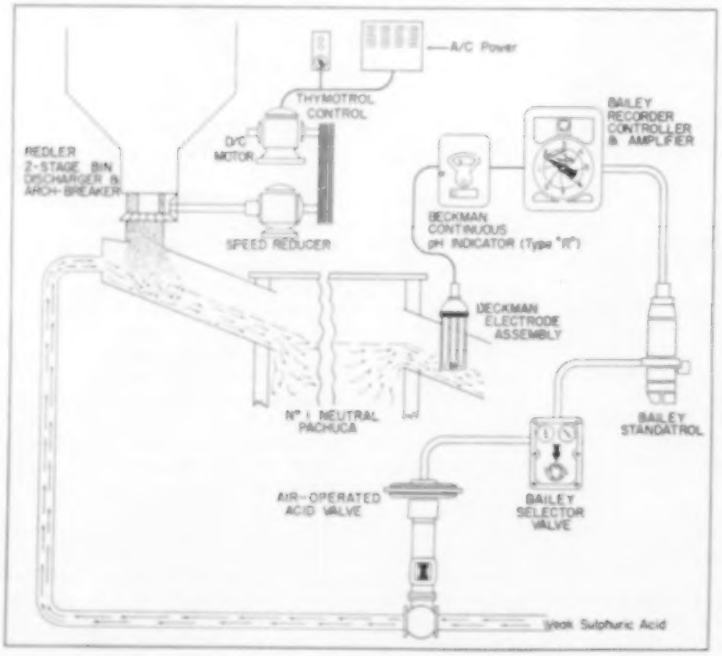

Fig. 7—No. 1 Pachuca pH control assembly has been drawn schematically and shows the first step in the automatic control of the neutral leach circuit. The pH control system was installed in order to prevent a large increase of impurities, i.e., arsenic, antimony, and germanium, in the neutral leach solution. Such an increase seems to be indicated when there is a pH of less than 5.0.

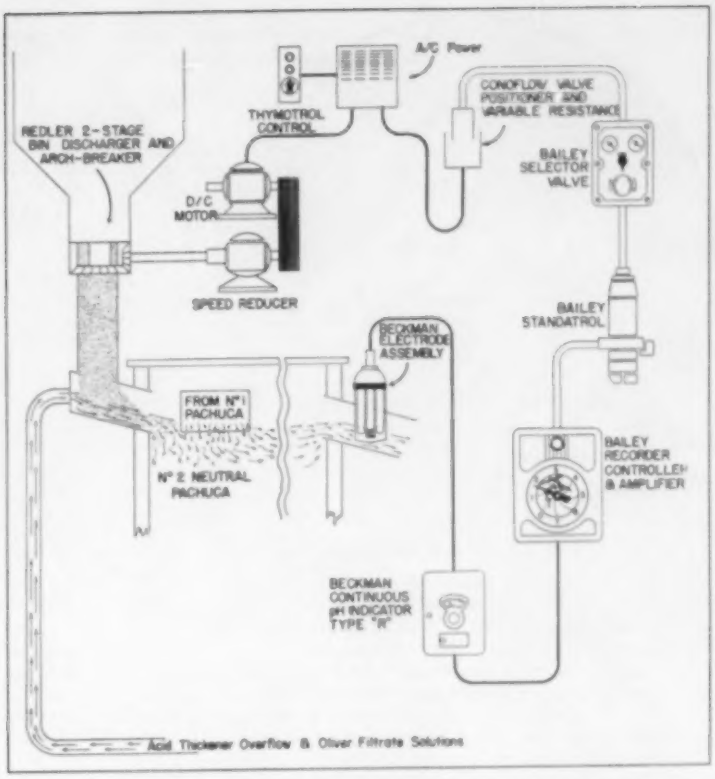

Fig. 8—Schematic drawing of No. 2 Pachuca pH control assembly illustrates the second pH control step. This is independent of step 1, shown in Fig. 7. Control on No. 2 Pachuca acts as a safety measure, allowing maximum leaching in No. 1. This control gives an assurance that the discharge from No. 2 will be at a proper pH level for subsequent purification.

pH control assembly. This unit is manually adjusted to give any desired feed rate.

Conditions under which No. 1 Pachuca is operated have been determined by experience. The present plant practice of discharging the pulp at a pH of 1.8 enables at least 80 pct of the total oxide delivered to the leaching plant to be fed to this Pachuca. Consequently, the bulk of the feed is given maximum leaching time. At this pH, no precipitation of impurities takes place within the tank and the short retention time does not allow all the oxide to be taken into solution, some oxide passing on to No. 2.

To obtain automatic control over these set conditions, the desired oxide feed rate is selected manually using the Thy-mo-trol control. The Bailey controller allows a pH to be selected at which the pulp from No. 1 Pachuca is to be discharged (pH 1.8). Any deviation from this set value is transmitted by controlled air pressure to the air-operated acid valve and compensation in acid flow is made automatically to maintain this selected pH. In other words, the pH of the pulp and oxide feed rate are held constant, leaving the rate of acid addition as the only controlled variable.

Certain safety devices have been incorporated into the control equipment of No. 1 Pachuca to guard against failure. The pneumatic acid valve is spring loaded. If for any reason the control air supply to this valve should fail, the valve will snap shut and an alarm bell will ring. The oxide feeder motor drives a hydraulic tachometer. Pressure exerted by the tachometer is transmitted to a solenoid valve. If the feeder motor should stop for any reason, the hydraulic pressure will drop to zero, the solenoid valve vents the air line supplying the acid valve to atmosphere, and the valve will snap shut. If the pH of the pulp discharge should deviate ±0.5 units from the set value, an alarm bell will ring and panel lights will indicate whether the deviation is either high or low.

Fig. 8 shows the second pH control step. Here, a pH electrode assembly dips into the pulp discharging from No. 2 neutral Pachuca. The pH of the pulp is continuously measured with a type "R" Beckman indicator. Again, the electrical impulse is amplified, recorded, and converted to an air pressure in a Bailey controller. Air pressure variations, corresponding to pH variations, are relayed through a Bailey Standatrol unit to a Conoflow valve positioner and variable resistance. Variations in resistance, which now correspond to pH variations, are used to control the oxide feed rate to this tank through a Thy-mo-trol motor control unit. The Bailey selector valve allows the equipment to be operated manually.

Operating experience has shown that No. 2 Pachuca should be discharged at a pH of 4.8. At this acidity, the pulp carries about 10 to 15 pct excess zinc oxide. The selection of this value allows the remaining three neutral Pachucas to operate in the pH range of 5.0 to 5.4, ensuring sufficient time for the precipitation of such dissolved impurities as iron, arsenic, antimony, and germanium. A pH lower than 4.8 would cause impurities to appear in the final neutral Pachuca discharge, whereas a higher pH would introduce too large an excess of zinc oxide in the final pulp.

To place No. 2 Pachuca under automatic control, the Bailey controller is set to a pH of 4.8. Deviations from this set value are transmitted by controlled air pressure to the variable resistance, and compensations in oxide feed rate are automatically made to maintain this selected pH. The pulp entering this

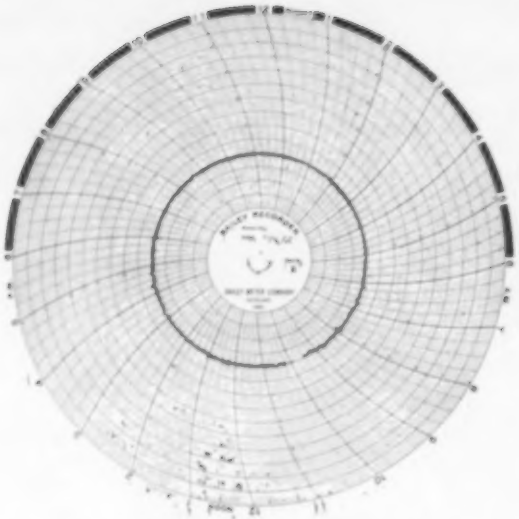

Fig. 9—Chart of the pH recorder shows the discharge from No. 1 neutral Pachuca.

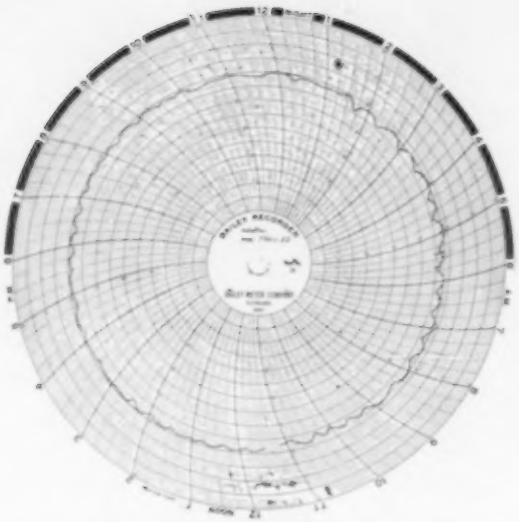

Fig. 10—Chart of the pH recorder shows the discharge from No. 2 neutral Pachuca.

tank is that which is discharged from No. 1 Pachuca and is at a constant pH of 1.8. Acid thickener underflow and residue filtrate solution that carries the oxide into this Pachuca are also at a constant pH of 3.5. Since the pulp discharging from this Pachuca is set at a pH value of 4.8, the only controlled variable in this step then is the rate of oxide addition.

The use of a two-step control system was chosen to minimize the danger of acid appearing in those tanks designated for purification (i.e., Nos. 3, 4, and 5). The control on No. 2 is actually a safety measure to allow maximum leaching in the first Pachuca with an assurance that the discharge from the second will always be at a proper pH level for later purification.

The automatic control equipment was designed and its installation supervised by the Bailey Meter Co. Ltd. Up to the present time, little trouble has been experienced with the equipment and shutdowns have been infrequent and of short duration. The conditions under which the electrode assembly must operate require daily maintenance of these units. They are cleaned by immersion in hydrochloric acid solution and then thoroughly washed with water. This procedure removes any incrustation of foreign matter. The life of the calomel electrodes is indefinite, since they can be renovated with fresh solution. Glass electrode life is about one month, failure being due to the abrasive action of the pulp on the glass envelope.

The final neutral Pachuca pulp, containing 3 pct solids, is discharged by launder to a Dorr-type DSFA classifier. Classifier slimes flow by launder to four Dorr torque thickeners. The thickeners, 50 ft diam by 15 ft, are of wood stave construction. All immersed metal parts are of Everdur. Clarified overflow from these thickeners leaves the plant by launder to the first of the neutral sulphide leaching plant Pachucas and represents 45 to 48 pct of the combined flow. An average assay of this solution appears in Table III. The neutral thickener underflow, carrying 30 pct solids, is pumped by 2 in. Dorr diaphragm suction pumps to the acid leach section of the plant. These pumps, as well as the lines to them, are made of Everdur.

The use of automatic control in the neutral leach stage has greatly simplified operations by ensuring consistent leaching conditions. Figs. 9 and 10 show the charts of the pH recorders for the discharge from Nos. 1 and 2 neutral Pachucas. This close pH control has resulted in a constant low impurity tenor in the neutral thickener overflow, constant excess of zinc oxide in the final Pachuca discharge pulp, and good settling qualities in the thickeners. The rate of iron addition is the only important manual operation

Table III. Oxide Leaching Plant Assays

Solutions	G per Liter					Mg per Liter					
	H_2SO_4	Zn	Cu	Fe^{++}	Fe^{+++}	As	Sb	Ge	F	Co	SiO_2
Oxide plant neutral thickener overflow	—	126.4	0.16	Trace	0.013	1.0	2.6	0.4	5.0	8.3	—
Oxide plant acid thickener overflow	—	125.2	—	Trace	0.033	9.7	8.4	2.7	5.9	—	319
Cell acid	106.9	54.5	—	—	—	—	—	—	—	—	—
Iron leach solution	37.3	58.3	—	0.7	13.8	176	53	—	—	—	—

Material	Pct Dry Weight										
	Total Zinc	Water Soluble Zn	Acid Soluble Zn	Cu	Fe	Pb	As	Sb	Ge	F	Insoluble
Oxide residue	19.8	8.9	4.3	0.7	13.4	10.6	4.6	1.0	0.016	0.063	13.00

Table IV. An Assay Comparison of Zinc Cell Electrolyte Produced Before and After the Advent of the Combined Flow Method of Processing Zinc-Bearing Materials

Materials Treated	Year	Zn, G per Liter	Mg per Liter Impurity							
			Co	Cd	Fe	As	Sb	Ge	F	Cu
Zinc sulphide concentrates only	1948	140.2	9.6	0.5	22.0	0.02	0.02	—	4	0.94
	1949	131.1	7.5	0.5	21.0	0.02	0.02	—	4	1.0
	1950	133.0	7.7	0.3	19.2	0.02	0.02	—	4	1.3
Zinc sulphide concentrate and zinc oxide fume	1951*	136.7	7.3	0.4	18.0	0.02	0.02	0.008	6.5	1.2
	1952	126.0	7.2	0.4	16.8	0.03	0.02	0.008	7.3	1.2
	1953	132.5	7.9	0.4	17.5	0.03	0.025	0.008	4.6	1.3

* Beginning May 1951.

necessitating close scrutiny by operating personnel.

The cell acid used in the oxide leaching plant is produced during the electrolysis of zinc sulphate solution. From the electrolytic cell room, the acid is laundered to 40 ft diam by 14 ft steel or wood stave storage tanks. All wood stave tanks are lead lined while the steel ones are lined with rubber. From storage, the acid is pumped through lead or polyethylene lines to the oxide leaching plant using 3x2 Worthite pumps. The acid is delivered to the first neutral Pachuca, the first acid leach tank, and to the iron leach tanks. Acid flow is considerably greater than plant requirements, ensuring available acid at all times to each operating unit. Excess flow is returned by launder to storage. The average assay of this acid appears in Table III.

Acid Leach Process: The acid leach section is comprised of three mechanically agitated rubber-lined steel tanks, 10 ft diam by 12 ft. Continuous leaching is made possible by connecting the three tanks in series by means of launders. Average solution flow through this section is 350 water tons per day.

Classifier sands from the neutral leach section are flushed through a launder to the first acid leach tank and are leached by a continuous flow of cell acid. The overflow from the first tank joins the neutral thickener underflow in a common launder and is fed to the second tank. Overflow from the second passes on to the third. Pulp discharge from the third tank, containing 10 pct solids, is laundered to two Dorr thickeners. These thickeners are similar in all respects to those in the neutral circuit.

Cell acid is added continuously to the first acid leach tank, the rate of flow being controlled by a manually operated acid valve. The flow is adjusted so that the pulp discharging from the third or final tank has a pH of 3.5. This value has been found to give maximum zinc recovery with minimum impurity pickup. A lower pH than this would introduce excessive amounts of impurities such as arsenic, antimony, and germanium into the neutral leach circuit by way of acid thickener overflow return. A portable Beckman pH meter is used for this control, and pH readings are taken every 15 min.

Acid thickener overflow, containing less than 0.1 pct solids, flows by gravity to a surge box. From here, it is pumped through copper lines by 4x3 Allis-Chalmers pumps to two 20 ft diam by 20 ft rubber-lined stave tanks. These tanks are located in the neutral leach section of the plant. Rate of discharge of this solution to No. 2 neutral Pachuca is controlled by manually operated valves.

Acid thickener underflow, containing 35 pct solids, is pumped by 2 in. Dorr diaphragm pumps to a common launder in which the pulp is carried to a mechanically agitated surge tank. This tank is of wood stave construction and is 14 ft diam by 10 ft. From the surge tank the pulp is pumped by a 3x2 Vacseal pump to two Oliver continuous drum filters. The level of the pulp in the filter trough is held constant by adjustable overflow weirs. Excess feed to the trough is returned to the surge tank.

Due to the slimy nature of the acid thickener feed, the settling qualities are poor. To obtain a relatively clear overflow, it has been necessary to withdraw about one-third of the flow to the thickeners as underflow. Consequently, the percentage of solids in the underflow is low, thereby adding to the difficulty of filtering the slimy pulp. Settling aids have been tried but, to date, none have been beneficial in the pH range at which the thickeners operate.

Each of the 8x16 ft Oliver filters have an effective filtering area of 400 sq ft. The drum surface is of wood stave construction, the staves being attached to rubber-covered cast iron drum heads. Wood staves are of British Columbia fir; drainage screens are of redwood. All internal drum piping is of copper and pipe fittings are of bronze. Drum speed is one revolution in 6 min, giving a total pickup time of 2 min per cycle. The filter cloth is monofilament Saran-type MS-942. The filter cake is discharged by wire scraper and falls into a rubber-lined repulp trough. Water sprayed through overhead wide-angled nozzles washes the cake while on the drum. No attempt is made to separate the pulp filtrate and wash water. Both are withdrawn from the filter through the same pipe. The filtrate is drawn into a rubber-lined steel receiving tank from which it is pumped by 3x2 Olivite pumps to the acid thickener overflow surge box. Vacuum for filtration is supplied by Ingersoll-Rand pumps at 18 in. Hg.

In the repulp trough, water is mixed with the filter cake to form a pulp containing 50 pct solids. From here, the slurry flows by gravity to a mechanically agitated surge tank, 8 ft diam by 8 ft.

Filtering difficulties, due to the slimy nature of the acid thickener underflow, have been overcome by the use of an open weave monofilament Saran cloth. Selection of this filter medium resulted from testing numerous types of cotton, Orlon, Vinyon, and Saran fabrics. The cake does not cling to the smooth Saran fiber. This reduces blinding to a minimum and aids in cake discharge. The filtrate carries about 1 pct fine solids that have passed through the cloth. This is not a great disadvantage, since the filtrate is returned to the leach circuit. There is no evidence of a build-up of fines in either the neutral or acid stages. Filter cloth life, under continuous operation, is about four months.

The cake formed on the filter drum is about 3/16 in. thick and very compact. Under these conditions,

Table V. Assays of the Two Grades of Slab Zinc Produced at Flin Flon, Manitoba

Grade	Pb, Pct	Cu, Pct	Cd, Pct	Fe, Pct	Zn, Pct
Four Nines Plus	0.0017	0.0021	0.0004	0.0004	99.9954
Electrolytic	0.0051	0.0024	0.0004	0.0013	99.9908

it is difficult to obtain a good wash. Soluble zinc losses are relatively high. To improve recovery, a third Oliver drum filter has been installed recently and is used to filter the repulped cake from the first or primary filtration step.

Primary repulped filter cake is pumped from storage to an 8x16 ft Oliver continuous drum filter using a 3x2 Vacseal pump. The filter is similar in all respects to the two primary filters. Saran-type MS-942 fabric is again used as a filter medium. Water is used for cake washing and is sprayed through overhead wide-angled nozzles. A wire discharges the secondary cake to a mechanically agitated repulp trough. Here, water is mixed with the cake to form a pulp carrying 50 pct solids. The pulp flows by gravity to a mechanically agitated surge tank 8 ft diam by 8 ft. At intervals of about 4 hr, the pulp is pumped to a storage pond using a 5x3 Wilfley sand pump. The 3 in. Transite line to the pond is 2150 ft long and is totally enclosed with a steam tracer line in an insulated wooden box. This arrangement is necessary to prevent freezing of the line during the winter months. Oxide residue production averages 24 dry tons per day, the assay of which appears in Table III.

Although the secondary filter is in operation, final plans have not been completed. Equipment is to be installed shortly to use the secondary filtrate as wash water for the primary filtration step. This arrangement will minimize water intake to the leaching circuit.

Floor drainage from the neutral and acid leach sections are kept separate. Each has its own collecting sump. Drainage from the neutral sump is pumped to the neutral thickeners; from the acid section sump, the drainage is pumped to the acid thickeners.

Revisions of the sulphide leaching plant were necessary in order to accommodate the increased flow resulting from processing zinc oxide fume. The number of neutral Pachuca tanks was increased from 9 to 11. Continuous zinc dust purification facilities were extended by 25 pct. Two additional cell acid storage tanks were installed, bringing the total to five. Finally, zinc dust consumption in the purification section was increased from 4½ to 5½ g per liter of solution flow.

It may be of some interest to compare the purity of the final zinc electrolyte produced before and after the advent of the oxide leaching plant and the method of *combined flow*. Prior to May 1951, only zinc sulphide concentrates were processed in the zinc plant. Some doubts were expressed as to the success of the combined flow method of treatment. The main reason for this view was that the high impurity content of the fume would cause a gradual build-up of harmful impurities such as germanium, antimony, and fluorine in plant solutions. As will be seen in Table IV, this build-up has not taken place. Cell room operations have remained normal in most respects. Current efficiencies in the zinc cells and the purity of metal produced have remained average.

Zinc metal production for 1953 averaged 180 tons per day. Two grades of zinc were produced during that year. The high purity grade Four Nines Plus was cast at the rate of 120 tons per day. *Electrolytic* or second-grade metal was produced at the rate of 60 tons per day. Average assays of both grades appear in Table V.

Acknowledgment

The writer wishes to thank the management of the Hudson Bay Mining and Smelting Co. Ltd. for permission to publish this paper.

Discussion of this paper, if any, will appear in JOURNAL OF METALS, November 1956, and in AIME Metals Branch *Transactions*, Vol. **206**, 1956.

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Technical Note

Method of Measuring Film Thickness of Solid Lubricants

by H. M. McCullough and I. Sheinhartz

DURING the course of a lubrication study in relation to the compacting of metal powders, it became desirable to measure the thickness of sprayed graphite coatings on steel dies for hot pressing. The thickness of these lubricant coatings ranged between 0.0001 and 0.002 in. In their particular case, the authors were interested in a method for measuring the lubricant thickness on tapered die inserts having curved surfaces. The lubricant was sprayed on the inserts at an elevated temperature and it was desirable to measure the thickness as soon as possible after application.

Various standard types of commercial equipment were tried. Most of the available devices operated on either a mechanical or magnetic principle. The mechanical devices, such as micrometers, dial indi-

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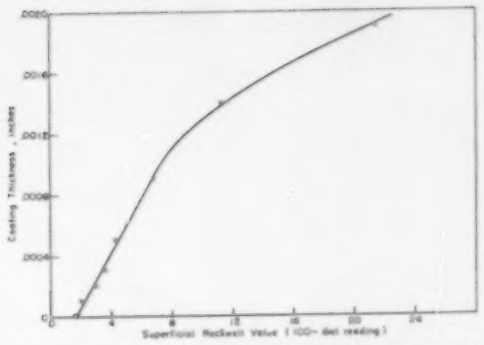

Fig. 1—Calibration curve for graphite coating thickness is given. Each point plotted is an average of four readings.

cators, shadow gages, etc., would be satisfactory if all the surfaces involved in the measurement were flat and parallel. The film thickness then could be obtained by measuring the overall thickness of die plus lubricant coating and subtracting the die thickness. The accuracy of these methods would depend on the dimensional variations, the parallelism of the die surfaces, and thermal expansion variations.

Magnetic devices are independent of the die geometry and they give a very sensitive direct reading. A serious drawback of the several magnetic devices that were tested was their sensitivity to oxide films. In the case of hot-pressing dies, the oxide film that was always present on the surface made it virtually impossible to get reproducible results of sufficient accuracy. Since dies were coated at slightly elevated temperatures, the variance in magnetic characteristics of the die materials upon heating also resulted in poor reproducibility of measurements. These methods are also limited to magnetic die materials. For the authors' purpose, none of the foregoing techniques was entirely satisfactory.

One other device that does not fit into either the mechanical or magnetic classification is the air gage. This method was used quite successfully to measure the coating thickness in cylindrical dies. The procedure is described in detail by N. P. Pinto.[1] This method is limited to cylindrical shapes, and the measurement has to be made at room temperature.

A very satisfactory device for measuring these graphite coatings on come-apart dies of any shape was found to be the superficial Rockwell hardness tester. This equipment is available in most laboratories, and it was found to have the necessary sensitivity and reproducibility. The measurements were made by using the standard operating procedure specified for the instrument. The coated die was placed on the anvil with the coating toward the penetrator. The loads were applied in the prescribed manner and a reading taken from the dial.

The ball penetrator used in the test does not actually penetrate through the graphite film but forms a shallow impression in the graphite. The ball compresses the graphite to a density high enough to support the major load that is being applied by the tester. The greater the thickness of the coating, the higher will be the compression needed to achieve the necessary density of the lubricant. The effect is similar to the pressing of metal powders where the higher the column of powder the greater the stroke necessary to attain a given density. Thus, the vertical motion of the penetrator is related directly to the coating thickness. Since each scale division represents 0.001 mm (0.00004 in.) vertical motion of the penetrator, the readings would be quite sensitive.

One requisite for the test to be satisfactory is that it be independent of the base material. This is necessary because the dies usually vary in hardness after being heated through several hot-pressing cycles. With the smallest standard load of 15 kg and a ⅛ in. diam ball penetrator, it was found that considerable variation in the hardness of the base material did not affect the readings that were obtained. This indicated that there was no penetration into the steel that the instrument could detect. The use of higher loads, or smaller diameter penetrators, yielded results that were more sensitive to hardness variations within the die itself. The effect of the base material on the test can be determined by making periodic readings on the uncoated steel. These readings should not vary.

The hardness tester was calibrated by measuring various coating thicknesses on a flat steel plate by means of a shadow gage and plotting these values against the equivalent readings obtained with the hardness tester. The shadow gage was set up with gage blocks and the results were very accurate. The resulting calibration curve is shown in Fig. 1. The results were reproducible within one-half scale division on the superficial tester. Each point plotted in Fig. 1 is an average of four readings, and the spread was never more than one division. This particular calibration curve would hold only for a particular type of coating, in this case graphite. Other materials would yield a different curve. Also, different types of indenters would give different curves. The nonlinearity of the curve shown is a typical curve from a ball-type penetrator.

The scale on the superficial tester is divided into 100 divisions numbered in a clockwise direction. The zero point would be the reading obtained on the base material without a coating. In this test it was always slightly to the left of the scale zero, i.e., a dial reading of 98 or 99. As the thickness of the coating increased, the dial reading would decrease in numerical value. In order to reverse this condition and get a value that increased as the coating thickness increased, the values plotted on the abscissa of Fig. 1 are equal to 100 minus the reading registered on the dial.

In these particular experiments, since the surface to be measured was concave, a ball penetrator had to be used. In the case of flat surfaces, it may be desirable to use rod-shaped penetrators having flat ends with the flat end facing the coating to be measured. This would decrease the sensitivity of the measurements in the case of thin coats and also make the test less sensitive to variations in the hardness of the base materials in cases where softer materials were being used.

In summary, the results indicate that the superficial Rockwell hardness tester can be used as a device for measuring the thickness of sprayed graphite and similar coatings at normal and slightly elevated temperatures. With a 15 kg load and a ⅛ in. diam penetrator, a minimum film thickness of about 0.0001 in. can be measured with an accuracy of approximately ±0.00002 in. The maximum measurable thickness would depend on the load and the diameter of the penetrator. It would increase as the load increased and the diameter of the penetrator decreased.

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Metallic Materials Resistant to Molten Zinc

by Webster Hodge, R. M. Evans, and A. F. Haskins

Refractory boron compounds are shown to resist corrosion by molten zinc. Coatings were made from ferroboron and manganese boron by several methods: welding, hard facing, and pack diffusion; and techniques of coatings are very important. Mechanical failure of the diffusion coatings can be partially eliminated by applying them to type 416 chromium steels rather than carbon steels. Welded coatings made with a tungsten arc are better than those made by other welding methods. Sintered compacts of mixtures of iron and chromium borides developed strengths of about 30,000 psi. They resisted corrosion of zinc at 600°C and oxidation at higher temperatures.

MOLTEN zinc rapidly attacks common metals of construction. The rate of attack is greatly amplified at temperatures over 500°C. However, there are many applications where it is necessary, or at least advantageous, to use metal construction, such as for galvanizing pots and zinc smelter equipment. Condensers and agitator linings used in continuous zinc smelting processes are eroded and corroded by violently agitated molten zinc. Pumps, tubing, and other parts to permit transfer of molten zinc would be desirable in many zinc handling operations. The development of metallic materials which are resistant to molten zinc also might permit its use as a heat-transfer medium in modern high efficiency power plants.

In reviewing previous work on zinc-resistant metals, it was evident that most of the available data refer to corrosion at temperatures just above the melting point. Tungsten and high Mo-Fe alloys containing more than 80 pct Mo resist corrosion by molten zinc.[1, 2] Imhoff states that because chromium is not attacked by zinc, chromium plating improves galvanizing pot life.[3] Nitrided cast iron has been used in die-casting machines[4] where the molten alloy is 96 pct Zn and 4 pct Al. Other metals that have been suggested as of possible value for zinc resistance are columbium, tantalum, and titanium.[5] The corrosion of steel by zinc has been discussed by many authors.[6, 7, 12] None of these materials resisted corrosion by molten zinc at the temperatures used in this investigation, i.e., 600° to 700°C.

I. Exploratory and Resistant-Coating Tests

To make a study of zinc-resistant metals, exploratory tests were conducted to sort out roughly those materials which seemed most worthy of further development. This preliminary study included both solid metals and surface coatings.

Exploratory Corrosion Tests in Molten Zinc—In initiating a study of this problem, numerous samples of metals and alloys were evaluated by dynamic corrosion tests made at 440°C. Each metal was tested in its most readily available form, which, because of shape and size, was often unsuited for standard corrosion testing.

The equipment used for dynamic corrosion testing of all materials, except in a few initial experiments, is shown in Fig. 1. Samples were tied to tungsten rods with tungsten wire, immersed in the zinc, and revolved at 12 rpm about the circumference of a 4 in. diam circle. Speed of travel was about 12.5 fpm. The molten zinc was protected from oxidation by an atmosphere of carbon monoxide, or, at a later period in the program, by a 9:1 mixture of nitrogen and hydrogen. When the effect of both molten metal and atmosphere was desired, the zinc level was lowered. Special high grade zinc was used in all of the corrosion tests.

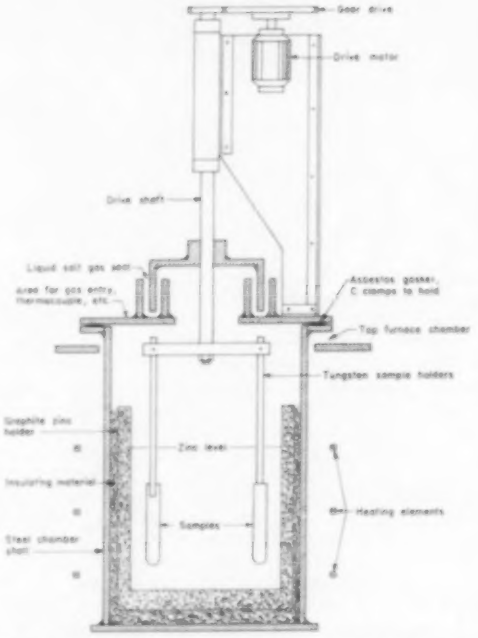

Fig. 1—Schematic diagram shows machine used for dynamic zinc corrosion testing of all materials except those used in a few initial experiments.

Corrosion losses for materials tested are shown in Tables I and II. Whenever the materials were available in suitable form, the losses are reported in mils per year. It was recognized that weight losses on irregularly shaped specimens did not show the corrosion rate but the important consideration—poor resistance to corrosion—could be determined readily.

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Discussion of this paper, TP 3996D, may be sent, 2 copies, to AIME by Sept. 1, 1955. Manuscript, Aug. 26, 1954. Chicago Meeting, February 1955.

Table I. Results of Zinc Corrosion Tests

Metal	Nominal Composition, Pct, and Condition	Pct Wt Loss in 50 Hr	Comments
	Commercial Alloys Tested at 440°C		
Stainless 446	27 Cr steel; sheet	22	—
Stainless 310	26 Cr, 20 Ni steel; sheet	—	Corroded too badly to weigh
Hastelloy "B"	33 Mo, 7 Fe, balance Ni	58	—
Hastelloy "C"	20 Mo, 18 Cr, 6 W, 7 Fe, balance Ni	23	—
Stellite 21	27 Cr, 6 Mo, 2 Ni, 1 Fe, balance Co	16	—
Hoskins 10	37.5 Cr, 7.5 Al, balance Fe	6	Not uniformly wet
Colmonoy No. 6	66 Ni, 17 Cr, 4 B, balance Fe, Si, C; rod	45	—
Duriron	14.5 Si cast	14	
Cast iron	—	27	
Colmonoy WRC 100	16 Cr, 3.5 B, 20 W, balance Fe; rod	1.7	
	Commercial Alloys Tested at 700°C		
Colmonoy WRC 100	16 Cr, 3.5 B, 20 W, balance Fe, Si, C; rod	12	—
Carbon steel	1010 cold rolled	—	Completely consumed
	Pure Metals Tested at 440°C		
Tungsten	Commercially pure, sheet	0.007	Not wet, not discolored, average 2 samples
Molybdenum	Commercially pure, sheet	3.2	Not wet, not discolored, average 4 samples
Titanium	Pure, iodide, vacuum melted, cold rolled	—	Completely consumed in less than 50 hr
Zirconium	Commercially pure	60	Fish-scale surface after corrosion
Tantalum	Commercially pure	—	Broken sample prevented weighing, corroded when surface was abraded in zinc
Chromium	Commercially pure	2.4	Attack concentrated where holder rubbed sample
Cobalt	Commercially pure	—	Completely consumed in less than 50 hr
Boron	Battelle made, pure	1.600	Weight loss due to fracture of sample, not wet
Silicon	Commercially pure	0.200	Weight loss due to fracture of sample, not wet
Manganese	Commercially pure, electrolytic	—	Completely consumed in less than 50 hr
Columbium	Commercially pure, foil	—	Completely consumed in less than 50 hr
Vanadium	Battelle, contained 3.5 pct Fe	49	—
	Pure Metals Tested at 700°C		
Tungsten	Commercially pure, sheet	0.04	Not wet
Chromium	Commercially pure	—	Consumed in less than 50 hr
Molybdenum	Commercially pure	12	
	Special Alloys Tested at 440°C		
80 Mo-20 Fe	Battelle made, sintered bar	0.4	—
95 Mo-5 Ta	Battelle made, melted button	0.02	Not wet, same results with sintered bar
80 Mo-20 Cr	Battelle made, melted button	—	Not wet, weight gain at 50 hr, same results with sintered bar
95 Mo-5 Cb	Battelle made, sintered bar	0.44	Wetted by zinc
$MoSi_2$	Battelle, Si vapor coated on Mo wire	0.14	Not wet, weight gain at 200 hr
Boronized steel	Battelle, B vapor coated on steel	6	Brittle coating showed craters after corrosion
Silicon iron	Coating from siliconized iron	40	

For discussion purposes, the metals listed in Tables I and II are divided into three groups: pure metals, commercial alloys, and special alloys.

Commercial Alloys: At 440°C, cast iron,* high silicon iron,* and mild steel* were almost as resistant to zinc as were the higher alloy metals. Alloys not resistant at 440°C were presumed not to be resistant at 700°C.

* It is common knowledge in the zinc industry that these metals corrode very rapidly in zinc at temperatures above 500°C.

Pure Metals: Tungsten, in a reducing atmosphere, was almost completely resistant to zinc, even at high temperatures. It did not appear to be wet by the zinc, either when abraded or vibrated ultrasonically in the molten zinc bath. Corrosion did occur at a very slow rate at the zinc atmosphere interface; two sets of ¼ in. diam W holding rods sufficed for almost continuous corrosion testing for a period exceeding 3½ years. Tungsten, however, has poor resistance to oxidation, especially above 600°C.

Boron and silicon, not being wet by molten zinc, were completely resistant to it. The losses recorded were due to experimental difficulties with the brittle samples. These results led to an extended research on commercial ferroborons, manganese borides, and chromium borides. Ferrosilicons were not included in the development, because siliconized iron and high silicon cast iron showed little promise of resistance to zinc corrosion at elevated temperatures.

Special Alloys: Molybdenum alloys, made by powder metallurgy techniques, and some special boron-containing commercial alloys of the Colmonoy type were evaluated. It was found that most of the additions to molybdenum did improve its resistance to zinc. Poor fabrication characteristics and high costs discouraged further development of these alloys.

Cr-W-B alloys of the Colmonoy type showed promising resistance to zinc in initial corrosion tests. However, alteration of these alloys by the addition of other resistant metals failed to improve the basic alloy.

Intermetallic compounds of several of the transition metals, fabricated by powder metallurgical techniques, were found to resist corrosion by zinc. Their development is reported in the section of this paper entitled "II. Sintered Borides."

Production of Zinc-Resistant Metallic Articles—The most promising materials for further study were boron, silicon, chromium, tungsten, molybdenum, and the alloys of these metals. Other alloys capable of resisting zinc were Fe-B, Mn-B, and sim-

Table II. Results of Zinc Corrosion Tests

Metal	Nominal Composition, Pct, and Condition	Corrosion Rate, Mils per Year, 50 Hr Test	Comments
	Commercial Alloys Tested at 440°C		
Carbon Steel	1010 cold rolled	686	—
	Pure Metals Tested at 440°C		
Tungsten	Commercially pure	0.3	Data from Table I
Molybdenum	Commercially pure	67.0	Data from Table I
	Special Alloys Tested at 440°C		
Mo-W	90 Mo-10 W, hot-rolled sintered	10.0	Not wet
Mo-W	80 Mo-30 W, hot rolled sintered bar	0.0	Not wet
Mo-W	70 Mo-30 W, hot rolled sintered bar	2.6	Not wet
Mo-Fe	15 Mo-85 Fe, forged bar	319.0	—
Mo-Fe	35 Mo-65 Fe, cast ingot	363.0	—
Mo-Fe	50 Mo-50 Fe, cast ingot	486.0	—
W-Fe	17 W-83 Fe, forged bar	685.0	—
	Pure Metals Tested at 700°C		
Tungsten	Commercially pure	2.5	Data from Table I
Molybdenum	Commercially pure	727.0	Data from Table I
	Special Alloys Tested at 700°C		
Mo-W	80 Mo-20 W, hot rolled sintered bar	264.0	Corrosion not uniform

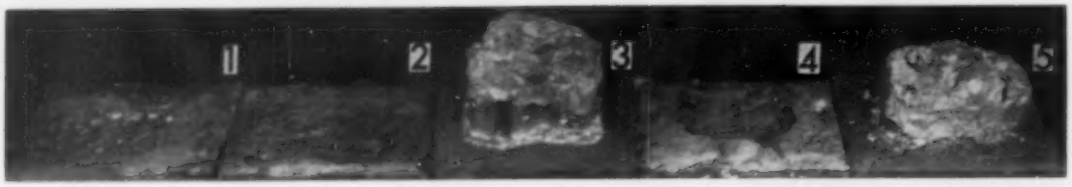


Fig. 2—Typical group of weld coatings of ferroboron on steel panels are shown after exposure to zinc at 590°C. Sample 1, normal coating produced by tungsten arc, showed serious attack after 100 hr exposure. Sample 2, thin coating produced by tungsten arc, showed very little attack after 100 hr exposure. Sample 3, extra thin coating, tungsten arc, failed after 40 hr exposure. Sample 4, extra heavy coating produced by tungsten arc, showed serious attack after 100 hr exposure. Sample 5, normal coating by carbon arc, failed after 40 hr exposure.

ilar borides and silicides. Most of these metals and alloys had some inherent characteristic which precluded their use in the available commercial form. Alloys containing significant amounts of boron, silicon, manganese, and chromium were hard, brittle, and unworkable. Alloys of tungsten and molybdenum required the development of special fabrication techniques. Diffusion coating, flame spraying, painting, welded surfaces, and powder metallurgy techniques were used in attempts to circumvent the difficult fabrication problems posed by the brittle refractory materials.

Coatings Produced by Arc and Gas Welding Methods: Zinc-resistant boron-containing alloy coatings were produced by welding boron or borides to steel with oxyacetylene, carbon arc, shielded tungsten arc, and atomic hydrogen arc. The coating alloy was applied either by use of especially produced welding rods or by coating a steel plate with a paste containing the alloy before welding. The coated samples were tested for resistance to corrosion by making the test plate the bottom of a refractory crucible containing zinc. Static corrosion tests at 590°C were used because of difficulties encountered in coating specimens on all sides, a necessity for dynamic corrosion testing.

Experiments with commercial ferroborons showed that a minimum of 12 pct B was required to give satisfactory resistance to molten zinc at 590°C. Therefore, the coating rods and pastes contained at least 18 pct B to help overcome dilution during welding. It was difficult to produce uniform coatings if the boron content of the rod or paste was increased to 24 pct. Coatings which were resistant to molten zinc at 590°C were produced by all weld surfacing methods, but consistently good coatings were obtained only when the shielded tungsten arc was used. Overheating and consequent dilution of the boron alloy with iron was a difficulty encountered with all surfacing methods.

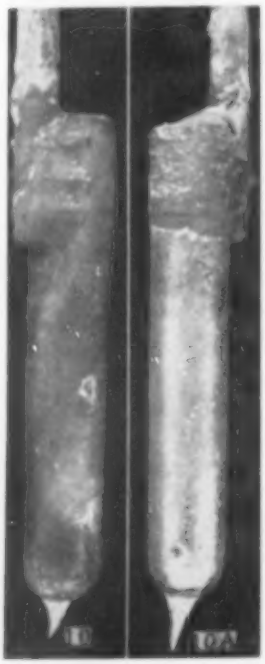


Fig. 3—Two specimens of steel, attached to small tungsten rods, were diffusion coated with pure boron and subjected to corrosion in zinc at 600°C. Specimen 10 was tested for 111 hr and 10A tested for 63 hr.

The best coatings were produced with the tungsten arc when the rods consisted of thin-walled steel tubes filled with 24 pct ferroboron. If a paste was used, 18 pct ferroboron with a small percentage of either tungsten or manganese added gave the best results. The binders used were important. Resinous binders were unsatisfactory because they carbonized and caused discontinuities in the coating. Silicate binders promoted the flow of the coating alloy; sodium silicate gave the best results with extruded rods, potassium silicate the best with paste coatings.

Fig. 2 is a typical group of weld-coated steel panels after exposure to molten zinc at 590°C. Varying degrees of *melting in* were used with all of the experimental coatings. It is evident from Fig. 2 that a better technique was required. Either too heavy or too thin coatings had less resistance than moderate coatings which are designated as *thin*.

Two types of attack by the zinc are evident in Fig. 2. On heavy coatings, corrosion is serious and uniform which indicates that too much melting in diluted the boron content below the required minimum. Too little melting in resulted in a pitting type of corrosion evidenced by the columnar structure of sample No. 3 in Fig. 2. The solid masses on samples No. 3 and 5 are refractory Zn-Fe compounds, corrosion products that were formed during the test.

These experiments showed that zinc-resistant coatings on steel can be produced by welding ferroboron of not less than the 18 pct grade to the surface of steel plates. Tungsten can be added to the ferroboron to increase the resistance of the coating to zinc. The development of the correct welding technique is very important for the successful production of zinc-resistant coatings.

Flame Sprayed, Painted, Vapor Deposited, and Electrolytic Coatings: Two types of flame spray guns were used to apply coatings to steel plates and rods; both guns sprayed the powdered boron alloy through an oxyacetylene flame. Coatings were made from 18 and 24 pct ferroboron and from commercial amorphous boron.

Uniformly adherent flame-sprayed coatings of ferroboron were difficult to apply because these coatings invariably spalled and crazed. Amorphous boron coatings rubbed off easily. Sintering of the

sprayed coatings improved the adherence of the coatings but did not prevent their spalling. Low melting Fe-B alloy formed during sintering caused washing. The resultant coating had areas that were too low in boron content for good resistance to zinc.

Painting powdered mixtures of boron or B-Fe alloys on a steel surface followed by heating proved impractical because a porous coating was left when the binders and carriers were burned out. Either fusing or sintering alloys painted on or applying them by flame spraying was not successful because a low melting low boron coating was formed.

Molybdenum, vapor coated with silicon, showed excellent resistance to molten zinc but the material had poor mechanical properties.

Vapor-deposited boron coatings were made on steel by exposure to boron trichloride at from 800° to 1000°C. Thin coatings made in this manner corroded uniformly in molten zinc; heavier coatings failed by pitting corrosion. Electrolytic deposits of boron on steel were not resistant to attack by zinc because the soft amorphous boron deposit was quickly eroded away.

Pack Diffusion Coatings: Pack diffusion coatings were produced by packing powder around an object to be coated and then heating the pack. Boron or boron alloys were diffused into steel samples at 980°C in an atmosphere of hydrogen or hydrogen chloride. The diffusion coating period varied from 18 to 48 hr. Diffused coatings were produced from pure boron, ferroboron (24 pct B), manganese (17 pct B) boron, chromium diboride, and mixtures of these materials. If properly applied, the coatings were almost completely resistant to molten zinc at temperatures up to 600°C. Dynamic corrosion tests that had a life of between 100 and 200 hr were common in the laboratory. Fig. 3 shows two specimens, attached to small tungsten rods, that were coated in pure boron. The bulged upper ends are formed by Sauereisen cement that was used as an aid in holding the specimens.

Early in the work on diffusion coatings with boron and ferroboron, it was found that the coating consisted essentially of two needlelike phases. If the second phase (high boron, *H* in Fig. 4) was not present, the coating was not resistant to zinc. The formation of the high boron phase depended upon sufficient length of coating time. If the coating times were too long, however, the coatings did not adhere well to the base material. Under these conditions, two types of cracks developed and caused subsequent failure. Peripheral cracks that formed between the outer zinc-resistant phase and the inner nonresistant phase (as between *H* and *G* in Fig. 4) caused the desired coating material to spall. Radial cracks also developed, as at *I* in Fig. 4 and *C* in Fig. 5, which permitted penetration of the zinc to the base metal. Figs. 4 and 5 are *before* and *after* pictures of the same specimen. In Fig. 5, *C* is zinc in a radial crack and *F* and *D* are high zinc areas. Area *E* in both figures is the base metal.

Cracking may be caused by differences in expansion of the various components of the coated steel. An examination of available literature on thermal expansion measurements revealed nothing on the subject of metal compounds applied as coatings on metals. However, the literature was replete with information on the coating of metals and ceramic bodies with glazes. The processes involved in these operations were applicable to the authors' problems.

The thermal expansion of the coating and base materials and their relation to each other are the most important of the factors that determine the adherence of a ceramic coating. Bell[14] states that the coating should be in compression (but only slightly) if resistance to cracking is to be expected. He does not clarify his reasoning that the coating is in compression if the coefficient of expansion of the coating is less than the base material. However, at high temperatures the coating material is relatively plastic so that, if the base material has the higher coefficient of expansion, the coating will be in compression when the composite is cooled. Dietzel and Meures[15] cite the need for a graduated series of layers in the coating with the expansion coefficient varying in order from the high value of the base to the low value of the outer coating. They also emphasize the role of coating thickness. Thinner coatings are considered best.

Merritt and Peters[16] and Saunders[18] show a method for using small chips of the coating material for interferometer measurement of the expansion. By adaptation of these methods, diffusion-coated steel tubes which were essentially all Fe-B compound were used to obtain expansion data on the coating material. Other works of Merritt and Peters[17-19] were also helpful in this work. Dilatometric measurements were made over the range from 20° to 600°C on the base steel and on a coated specimen, and interferometric measurements were taken on chips of the coating. These measurements produced the following data: carbon steel, 16.5×10^{-6} in. per °C; coated specimen, 13.6×10^{-6} in. per °C; and coating chips, 13.0×10^{-6} in. per °C.

Since the expansion coefficient of carbon steel is so different from that of the coating, cracks would be expected in the coating. To minimize this difference, 12 to 14 pct Cr steel with a coefficient of expansion of about 13×10^{-6} in. per °C was chosen. Diffusion coatings on types 416 and 430 stainless steels were similar to those on low carbon steels; see Fig. 6. Although the radial cracks were eliminated, the peripheral cracks remained.

Corrosion tests of boron coatings on chromium steels showed great improvement over similar coatings on plain carbon steels. Penetration of the zinc

Fig. 4—*Before*: micrograph of pure boron diffusion coating on steel before exposure to zinc shows that the coating consisted essentially of two phases, *H* and *G*; *I*, radial crack; *E*, base metal. Specimen as polished. X100. Area reduced approx. 45 pct for reproduction.

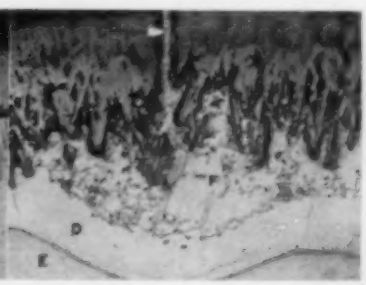
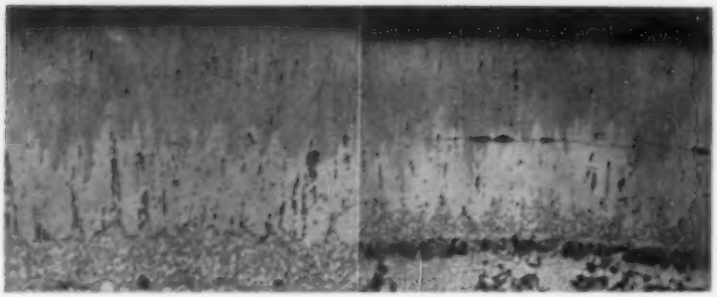


Fig. 5—*After*: micrograph of the same specimen as in Fig. 4 shows the effect of exposure to zinc. *C* labels a zinc-filled radial crack; *F* and *D* are high zinc areas. *E* is the base metal. Specimen as polished. X100. Area reduced approx. 45 pct for reproduction.

a—Type 416 chromium steel was treated for 36 hr.



b—Type 416 chromium steel was treated for 36 hr.

Fig. 6—Chromium steel samples were diffusion coated in pure boron. The microstructure studies show that radial cracks were eliminated but that peripheral cracks remained. Both samples were etched with picral. X100. Area reduced approximately 45 pct for reproduction.

through radial cracks to the base metal was eliminated. Spalling of the coating was responsible for all failures. Fig. 7 shows three chromium steel specimens which were exposed to molten zinc at 600°C. Two types of failure are apparent: pitting as illustrated in specimen No. 1 and spalling as in specimen No. 2. Specimen No. 3 shows a later stage of corrosion after spalling has occurred.

The pitting failure was rather uncommon and was associated with a honeycombed outer high boron phase that was sometimes obtained. Such a coating is shown in Fig. 8. The cause of the honeycombed structure was not found, but it is thought to be associated with such variables as rate of heating, atmosphere, and possibly density of the packed coating material.

Thin coatings, although they were more adherent, were not resistant because the high boron phase was not present. The addition of tungsten, molybdenum, and chromium to the coating powder mix failed to eliminate spalling and cracking.

Conclusions—No common metal or alloy has been found that is resistant to zinc and zinc vapor and to oxidation at 600°C. Tungsten resists corrosion by molten zinc at this temperature, but it fails in an even slightly oxidizing atmosphere.

Zinc-resistant coatings were found that could be applied to low carbon steels. These coatings can be made by welding and hard facing methods. Diffusion coatings on type 416 stainless steel showed special promise. Zinc-resistant coatings were made from refractory boron alloys such as ferroboron and manganese boron that contain at least 12 pct B. The technique used to apply the coating material was extremely important in the production of coatings free from defects. Consistently satisfactory coatings were difficult to obtain.

Basic knowledge of the resistance of various materials to corrosion by molten zinc has been gained from these experiments. More work is needed to develop further the practical equipment needed for handling zinc based on these findings. It is believed that additional study of the following factors may permit consistent production of crack-free zinc-resistant coatings: 1—effect of surface preparation, 2—effect of particle size in diffusion packing powder, 3—effect of atmospheres, 4—identification of metal phases produced during coating, and 5—choice of proper combinations of coating and base materials.

II. Sintered Borides

Coatings of iron boride and mixed borides of chromium or manganese and iron on steel have been shown to resist corrosion effectively in zinc or zinc vapor at temperatures up to about 600°C (1100°F), if only the higher boron phases of these metals are exposed to zinc. Furthermore, it was shown that FeB, the higher boron phase, could be diluted with iron to a mixture of Fe_2B and FeB containing only 12 pct B and still retain most of its resistance to attack by molten zinc. With this knowledge, it was a natural step to investigate the corrosion resistance of sintered materials whose composition could be controlled closely.

Some aspects of the sintered borides under consideration had been studied by Hägg and Kiessling[20] and McBride et al.[21] Steinitz and Binder[22] investigated sintered bodies formed by powder metallurgical techniques from borides of the transition metals. The ultimate objective of their work was to develop mechanically strong and oxidation-resistant materials suitable for cutting tools. To the authors' knowledge, no previous attempt had been made to develop these materials for resistance to corrosion by liquid metals. Recently however, Schwartzkopf and Glaser[23] have reported that sintered zirconium

Fig. 7—Three boron-coated chromium steels were exposed to molten zinc at 600°C. Specimen 1 shows pitting-type failure; specimen 2, spalling; and specimen 3 illustrates a later stage of corrosion after spalling has occurred.

boride shows high resistance to aluminum, tin, high copper brasses, lead, and copper.

Raw Materials— The raw materials used in this phase of the research were: iron boride (ferroboron) containing 24 pct B; manganese boride, 17 pct B; a nominal amount of chromium boride, 26 pct B; and various metal powders. The iron boride contained a large excess of boron over the amount required to form FeB. The manganese boride was essentially MnB. The chromium boride was a commercial product sold as "CrB_x". Although the actual composition of CrB_x may have varied appreciably from the theoretical formula, its exact composition was not determined. In any case, it contained an excess of boron beyond the amount required to form CrB. The inability to form strong sintered compacts with manganese boride and iron probably was influenced by the lack of an excess of boron in the base material over that required to form MnB.

Preparation of Compacts—All of the borides that were cold pressed were prepared with a binder for added green strength. The maximum pressure for compacting usually was 11,000 psi. The green compacts were dried at about 150°C before presintering.

Powders to be hot pressed were placed in the mold dry with no additional binder.

Sintering: A review of the Fe-B phase diagram may assist in understanding the reactions during sintering. At the high iron end of the diagram, a eutectic with 3.8 pct B that melts at 1174°C occurs between pure iron and the ϵ phase. The ϵ phase, which theoretically contains 8.82 pct B, melts at 1389°C. The diagram also shows a phase, η, containing 16.23 pct B. The η phase inverts to η' at 1131°C, which melts at approximately 1550°C. In this paper ϵ is referred to as Fe_2B and η or η' as FeB. Both manganese and chromium, with atomic weights close to that of iron, form similar compounds. The binary phase diagrams of B-Mn or B-Cr are very similar to the B-Fe diagram. Similar borides of these metals (e.g., FeB, CrB) formed solid solutions (mischcrystals) in every sample examined during this study.

Melting point determinations were made as a guide in selecting sintering temperatures. Normally for powdered metal compacts, the proper sintering temperature would be in the range from about halfway between room temperature and the melting point to 75° or 50°C below the melting point. In this work, it was found that rapid sintering at higher temperatures produced the best bars.

Compacts sintered slowly at lower temperatures formed sweat beads. Presumably the boron and iron first combine to make Fe_2B, which then forms eutectic with uncombined iron. The beads observed on slow sintering are composed of the eutectic. Later, when all of the iron has combined with boron, the eutectic disappears. Rapid high temperature sintering apparently forces the compound to form at a rate such that sweating of the eutectic cannot occur. All sintering and presintering operations were conducted in a hydrogen atmosphere.

Compacts based on iron boride could be sintered equally well whether they were heated in a muffle or by induction. Compacts based on chromium boride heated somewhat inefficiently by induction, while those based on manganese boride could not be brought to sintering temperature by this method. Manganese boride (MnB) appeared to act as a nonconductor.

Properties of Cold Pressed and Sintered Bodies: Typical properties of cold pressed and sintered compacts of uniform cross-section are shown in Table III. The data indicate the advantage in strength obtained when a part of the boride is formed by diffusion into metal powder admixtures.

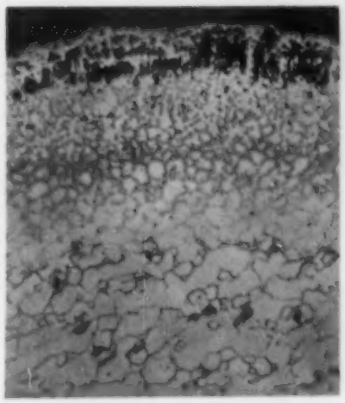


Fig. 8—Pitting failure of boron diffusion-coated type 430 stainless steel was rather uncommon and associated with a honeycombed outer high boron phase sometimes obtained. Specimen picral etched. X100. Area reduced approx. 40 pct for reproduction.

The least expensive mixture from which reasonably high and consistent strengths were obtained was 67.7 pct ferroboron (24 pct B) plus 32.3 pct Fe powder. Sintered compacts of this mixture should be essentially FeB. Compacts based on chromium diboride with additions of iron, cobalt, and nickel were stronger and more resistant to oxidation. The bar graph in Fig. 9 indicates the comparative strengths of the sintered borides and those of graphite and gray iron. Since cobalt and nickel are expensive, mixtures containing these elements would be chosen only when the density, increase in strength, and oxidation resistance over that of the CrB_x-Fe mixture might be especially important.

Effect of Reheating in Air: The strength of the 80 pct CrB_x-20 pct Fe compacts could be nearly tripled (from 10,300 to 29,900 psi) by reheating them to 700°C in air for 2 hr. Reheating similar compacts in air to 800°C increased their strength only slightly,

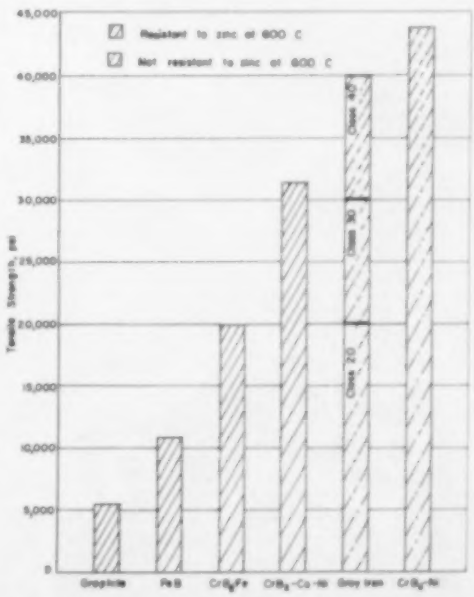


Fig. 9—Strengths of sintered borides are compared with those of graphite and gray iron.

Table III. Properties of Cold-Pressed and Sintered Compacts*

Powder Composition, Wt Pct		Compacting Pressure, Psi	Presinter		Sinter		Calculated† Tensile Strength, Psi	Density, G per Cu Cm
Boride	Metal		Temperature, °C	Time, Min	Temperature, °C	Time, Min		
Fe 100	—	11,000	—	—	1200	180	1,700	2.67
Mn 100	—	11,000	Could not sinter				—	—
Cr 100	—	11,000	Could not sinter				—	—
Fe 67.7‡	Fe 32.3	11,000	1435	15	1500	5	10,800	5.45
Mn 90§	Fe 10	11,000	1200	15	1420	15	2,600	—
Cr 85**	Fe 15	11,000	1200	15	1600	30	20,000	4.36
Cr 80	Fe 20	11,000	1200	15	1593	15 and 30	††	c. 5.6
Mn 90§	Co 10	11,000	Could not sinter in 10 min at 1620°C				—	—
Cr 95	Ni 5	11,000	Could not sinter in 10 min (temperatures to 1590°C)				—	—
Cr 95	Co 5	11,000	Could not sinter in 45 min at 1650°C				—	—
Cr 85	Ni 15	11,000	1200	15	1593	5	44,000	5.70
Cr 85	Co 15	11,000	1200	15	1593	15	14,900	5.86
Cr 85‡‡	Ni 5 Fe 10	11,000	1200	15	1593	15	17,700	5.43
Cr 80§§	Ni 5 Co 15	11,000	1200	15	1593	10	31,400***	6.13

* Specimens were 3.0x0.345x0.3 in. thick before sintering. After sintering 5 min at 1500°C, the dimensions varied considerably. Calculated volume shrinkage was 36 pct.
† Tensile strength was calculated from the modulus of rupture in transverse loading.
‡ Specimen intended to contain 16 pct B.
§ Specimen intended to contain 15.3 pct B.
** Specimen intended to contain 24.6 pct B.
†† Strengths varied widely, from 4,600 to 13,100 psi, irrespective of sintering time.
‡‡ Specimen intended to contain 24 pct B.
§§ Specimen intended to contain 23 pct B.
*** Strength of specimens varied from 27,600 to 38,800 psi and the average strength was 31,400 psi.

while reheating to 900°C in air decreased their strength. Reheating the CrB_x-5 pct Ni-10 pct Fe compacts in air for 2 hr at temperatures in the range from 700° to 1300°C had little effect on their tensile strength. Some gain in strength was obtained by reheating the CrB_x-15 pct Co-5 pct Ni bars for 2 hr in air at any temperature from 900° to 1300°C. Six bars, broken after reheating, had strengths averaging 34,000 psi. Individual bar strength varied after reheating from 30,500 to 38,100 psi. No change in dimensions or appearance could be observed in any of the bars after reheating.

These oxidation experiments proved that sintered bodies made of 80 pct CrB_x-20 pct Fe are resistant to oxidation up to somewhat above 700°C, and those made of 85 pct CrB_x-10 pct Fe-5 pct Ni or 80 pct CrB_x-15 pct Co-5 pct Ni are resistant up to at least 1300°C. Time was unavailable to determine the cause of the very impressive gain in strength of the CrB_x-FeB compacts after reheating them in air at 700°C. Possibly the compacts contained some Fe_2B which was converted to FeB by soaking at 700°C. At 900°C, some of the FeB may have been oxidized.

On sintering, cold-pressed compacts undergo a shrinkage slightly greater than one-third of the original volume.

Hot Pressing: Borides, like metal powders, may be hot pressed in graphite molds. The final compact thus obtained conforms to the mold dimensions.

Hot-pressing experiments were confined to iron boride (FeB) produced by diluting 24 pct ferroboron with iron powder. Cylinders ½ in. diam and 2 in. long were pressed in graphite molds that were heated in an induction furnace. A pressure of 4000 psi was applied momentarily by graphite pistons acting on each end of the cylinder when the temperature reached 1400°C. This pressure was released immediately and the compact was cooled in the mold. The tensile strength of two typical bars made in this way were, respectively, 15,000 and 10,700 psi. The corresponding densities were 5.42 and 4.97 and the apparent porosities 0.5 and 2.93 pct. These properties correspond closely to those obtained by cold pressing and sintering the same material containing an organic binder.

Production of Irregular Shapes—Although this research was confined to an evaluation of materials resistant to molten zinc, several attempts were made to produce shapes other than round or rectangular bars. Angle sections with one leg thicker than the other were hot pressed directly. Some excellent specimens were obtained, but it would be difficult to obtain long pieces by this method. One quite successful method of making irregular shapes, such as small impellers, was to cold press a solid compact of sufficient size and then carve or machine the compact to the desired form. The shaped part was presintered and then sintered as usual.

Small propellers and impellers were hot pressed in shaped graphite dies. This method was very successful, although some difficulty was encountered when a cored hole was required. When bodies were hot pressed around a graphite core, it was necessary to remove the graphite core at the same time that the pressure on the die was released. Otherwise, the part cracked as the material cooled and contracted. Figs. 10, 11, and 12 show small parts made by these methods.

During the attempt to produce irregular shapes by carving cold-pressed compacts, it was noted that better results could be obtained if the compacting pressure was increased beyond the 11,000 psi normally used. However, machining operations became more difficult with denser compacts.

Resistance to Corrosion by Zinc—The sintered iron boride compacts with 16 and 24 pct B were completely resistant to corrosion by molten zinc at 595°C. Similarly, complete resistance to corrosion was exhibited by the following pressed and sintered compacts: 85 pct CrB_x-15 pct Fe powder; 85 pct CrB_x-15 pct Co powder; 80 pct CrB_x-15 pct Co powder and 5 pct Ni powder; and 90 pct MnB-10 pct Fe powder. Compacts of CrB_x-15 pct Ni powder, which were the strongest achieved (see Table III), failed rapidly in molten zinc.

Sintering of MnB compacts was never completely successful, since even after long sintering periods the centers of the compacts remained soft. At higher temperatures the mixture melted. Addition of 10 pct Fe powder to the MnB did not aid sintering appreciably, and addition of cobalt made sintering still more difficult.

Hägg and Kiessling[9] reported that, in sintered mixed borides of the transition metals, the metal with the lowest atomic number is always richest in boron. Therefore, assuming that equilibrium was attained, the mixed borides formed in these experiments may be assumed to have approximately the composition given in Table IV. Among all of these compounds, only NiB,* Fe_2B, and Mn_2B are not resistant to molten zinc at 595°C. The NiB appears to have a particularly deleterious effect, since compacts containing only 17.8 pct NiB disintegrated in molten zinc in less than 24 hr. The compacts of MnB plus approximately 11 pct each of Mn_2B and Fe_2B resisted corrosion. A small amount of nickel (up to 5 pct) combined with CrB and CoB increased the strength of this type of compact without apparent damage to its resistance to corrosion.

* Time was not available to determine by X-ray diffraction if NiB was actually formed. Possibly only Ni_2B was formed during sintering. Even if this is true, the nickel compound is more undesirable than similar compounds of the other transition metals investigated.

Corrosion tests on these materials were made in the dynamic tester described previously. Specimens were moved continuously through molten zinc at 595°C for periods up to 400 hr. Within the limits of the tests, all specimens except those containing 15 pct Ni were completely resistant to corrosion. Thus, no relative rating of the materials is possible.

After continued stirring in the zinc bath, the pores of the compacts became filled with zinc. This absorbed zinc could be sweated and burned from the compacts without damaging them.

Conclusions—The compounds FeB, MnB, and CrB_2 resist corrosion by molten zinc at 595°C. Although it has not been tested alone, CrB probably also resists corrosion by zinc. In any case, the presence of some 16 pct CrB_2 or more than double that amount of mixed CrB and CoB, in a base of CrB_2, did not lower the resistance of the base material to corrosion at the test temperature. MnB, which completely resisted corrosion by molten zinc, could not be fabricated satisfactorily from the material available (MnB without excess boron) as a starting material.

NiB increases the strength of CrB_2 compacts, but NiB is not resistant to corrosion by zinc. Admixture of 15 pct Ni with CrB_2 destroys the corrosion resistance of the mixture.

Table IV. Compositions of Mixed Borides Formed in Tests

Original Mixture	Composition Obtained
CrB_2 + 15 pct Fe	65.2 pct CrB_2 + 16.9 pct CrB + 17.9 pct FeB
CrB_2 + 15 pct Ni	66.2 pct CrB_2 + 16.0 pct CrB + 17.8 pct NiB
CrB_2 + 15 pct Co	66.25 pct CrB_2 + 16.0 pct CrB + 17.75 pct CoB
CrB_2 + 15 pct Co + 5 pct Ni	55.0 pct CrB_2 + 21.3 pct CrB + 5.9 pct NiB + 17.75 pct CoB
MnB + 10 pct Fe	78.2 pct MnB + 10.8 pct Mn_2B + 11.0 pct Fe_2B

Recrystallization and growth of new borides by diffusion of boron from high boron compounds into most of the transition metals aid in deformation of strong bodies during sintering of powdered materials.

Cr-Fe-B compacts, formed by sintering eight parts of CrB_2 with two parts of iron powder for 15 min at 1593°C in hydrogen, then reheated in air for 2 hr at 700°C, developed tensile strengths of approximately 30,000 psi. Similar strengths and greater resistance to oxidation can be obtained by sintering CrB_2 with 15 pct Co and 5 pct Ni.

Fabrication of small articles indicates that useful products, such as pump parts, die casting machine parts,† and others, which are subject to severe corrosion by molten zinc, may be made of sintered powdered borides.

† Die casting parts are in contact with 96 pct Zn-4 pct Al alloys. At normal temperatures (400°C), this alloy is less corrosive to iron than is zinc. Even under these conditions, die casters report difficulty with corrosion of pistons and the rings through which they operate. The authors believe that sintered boride rings and pistons, operating in 96 pct Zn-4 pct Al alloy at 400°C, would resist corrosion indefinitely.

Some indication was obtained that MnB may be useful as a high temperature insulator.

Applications for patent rights covering the novel features discovered in the work on sintered borides has been made by the St. Joseph Lead Co.

Further development of sintered borides as useful materials to resist corrosion by liquid metals is warranted.

Acknowledgments

The authors wish to thank the St. Joseph Lead Co.

Fig. 10—Screw propeller was formed by shaping a green compact of powdered CrB_2 (85 pct) and iron powder (15 pct) and sintering after shaping. One blade was ground after sintering. The small crack between blades was caused by poor design. X2. Area reduced approx. 30 pct for reproduction.

Fig. 11—Impeller was formed by shaping a green compact of powdered CrB_2 (85 pct) and iron powder (15 pct) and sintering after shaping. The simple design is appropriate for the material used. X2. Area reduced approx. 30 pct for reproduction.

Fig. 12—Impeller was formed by hot pressing powdered ferroboron (85 pct) and iron powder (15 pct) to make FeB (16 pct B). X2. Area reduced approx. 30 pct for reproduction.

and Carleton C. Long of that company for permission to publish this information. Acknowledgment is also made of the helpful advice and encouragement offered by Dr. Long and by Bruce W. Gonser of Battelle Memorial Institute during the course of the laboratory work.

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Technical Note

Crystal Structure of TiRu and TiOs

by Charles B. Jordan

LAVES and Wallbaum[1] have described TiRu and TiOs as having the same structure as TiFe; they had previously described the latter compound as having the CsCl structure. On the other hand, a careful investigation of the structure of TiFe by Duwez and Taylor[2] failed to reveal the ordered CsCl structure, finding instead a random arrangement of titanium and iron atoms on the points of a body-centered-cubic space lattice. Consequently, it is not clear whether Laves and Wallbaum proposed the ordered or the disordered structure for TiRu and TiOs. In order to settle this question as to the structural type and to obtain values of the lattice parameters (thereby completely determining the structures), the present investigation was undertaken.

Samples having the compositions TiRu and TiOs were weighed out accurately from iodide titanium supplied by the New Jersey Zinc Co., ruthenium powder from Goldsmith Brothers Smelting & Refining Co., and osmium powder from Wildberg Brothers Smelting & Refining Co. No analysis was made of any of these metals. According to the respective analyses of the suppliers, total impurities amount to less than 0.03 pct in the titanium and to less than 0.5 pct in the osmium. The purity of the ruthenium is not known. The alloys were prepared by arc melting in a furnace of the type described by Schramm, Gordon, and Kaufmann.[3] Specimens for X-ray diffraction were obtained by crushing the ingots and passing the resulting powders through a 200 mesh screen. Diffraction patterns were made with a powder

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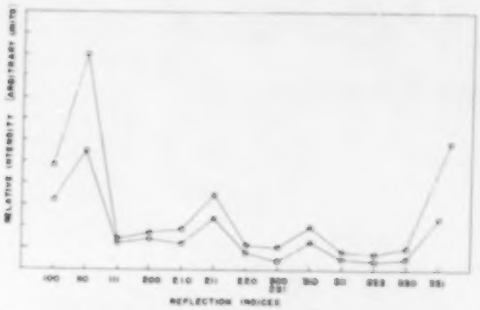

Fig. 1—Curves give comparison of observed (lower curve) and calculated (upper curve) intensities for TiOs.

camera 14.32 cm diam, the screened powders being irradiated with copper radiation filtered through nickel foil. Patterns were taken both before and after annealing the powders according to the procedures given in Table I. In addition, patterns of TiOs were made on a General Electric spectrogoniometer, with the Geiger counter scanning the complete pattern (made with filtered CuK radiation) at a speed of 0.2° per min.

All the diffraction patterns clearly showed both TiRu and TiOs to have structures based on a simple cubic space lattice and to have a unit cell edge of 3.06Å for TiRu and 3.07Å for TiOs. No attempt was made to obtain more accurate values of the lattice parameters. The lines of all the powder patterns of both substances clearly exhibited a definite alternation of intensity, that is, weak lines alternated with stronger lines in each pattern. Moreover, the weak set of lines in every instance corresponded to those reflections which would have been absent if the lattice had been body-centered-cubic.

Relative intensities were calculated for both TiRu and TiOs by taking into account the Lorentz, polarization, multiplicity, and structure factors and assuming the structures to be the *B*2 (CsCl) type (i.e., one titanium at 0,0,0 and one ruthenium or osmium at ½,½,½). The intensities thus calculated were not found to be in good agreement with those estimated visually from the photographically recorded patterns. Since this lack of agreement might be attributed to the effect of absorption, two methods for improving the agreement were tried. First, patterns of TiOs were made employing a General Electric spectrogoniometer, which is designed to use Bragg-type (rather than Laue-type) reflection and to reduce absorption effects to a minimum. Second, an absorption correction was calculated using the method of Bradley[4] for both TiRu and TiOs. Intensities were measured on the spectrogoniometer charts in two ways: 1—by measuring peak heights and 2—by measuring the area under each peak with a planimeter (both components of the Kα doublet being included for the higher angle reflections). No improvement in the agreement between calculated and measured intensities resulted from either the Bradley correction or the use of integrated intensities from the spectrogoniometer charts. Intensities as measured by peak heights on the spectrogoniometer charts were found to give best agreement with the calculated values. These calculated values did not include an absorption correction in order to obtain an appropriate comparison with values measured by the spectrogoniometer. When the peak height measurement was used, only one reflection, namely, (321), was found to be in significant disagreement with the calculated intensities. This disagreement is probably due to the separation of the Kα doublet for the (321) reflection; this separation would lower the intensity as measured by peak height. Fig. 1 presents a comparison of the relative intensities for TiOs as calculated and as determined from peak heights on the spectrogoniometer chart.

Table I. Preparation of Powders for X-Ray Diffraction

Substance	Recording Method	Condition of Powder
TiRu	Film	Not annealed
TiRu	Film	Annealed 24 hr at 1000°F
TiOs	Film	Not annealed
TiOs	Film	Annealed 24 hr at 1000°F
TiOs	Spectrogoniometer	Not annealed
TiOs	Spectrogoniometer	Annealed 24 hr at 1400°F and 120 hr at 1000°F

The nature of the powder patterns obtained for both TiRu and TiOs provides strong evidence that these substances crystallize in the *B*2 (CsCl) type of structure, despite the difficulty encountered in satisfactorily matching observed and calculated intensities. For a crystal of *B*2 type, the structure factors for all reflections (*hkl*) are given by

$$F(hkl) = f_a + f_b$$

when $h + k + l$ is even, and

$$F(hkl) = f_a - f_b$$

when $h + k + l$ is odd, where f_a and f_b are the atomic scattering factors for the two kinds of atoms in the structure. Since the sum $h + k + l$ is alternately even and odd for the reflections from a cubic crystal, it follows that the *B*2 structure gives rise to the same kind of pattern of alternating intensities as observed from both TiRu and TiOs. Moreover, the intensities of the weaker lines (i.e., those for which $h + k + l$ is odd) are greater as the difference $f_a - f_b$ is greater. Thus, if TiRu and TiOs have the *B*2 structure, the weaker set of lines in TiOs should be more intense than those in TiRu, in agreement with the observed fact.

The formation of TiRu and TiOs is accompanied by a decrease in the sizes of the atoms involved. The lattice parameters of the two compounds, as calculated from the Goldschmidt atomic diameters[5] corrected for the change in coordination number from 12 to 8, are 3.14Å for TiRu and 3.16Å for TiOs, as compared with the respective measured values of 3.06Å and 3.07Å. Thus, a shrinkage of about 2.5 pct is indicated.

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Experimental Planning for Rapid Determination Of Optimum Process Conditions

by W. A. Griffith

Fractional replication of factorial design, a general method for planning experimentation and for analysis of data obtained, is described as applied to a flotation investigation. This method leads to determination of optimum process conditions with minimum experimental effort. Its advantages over simple factorial design are demonstrated.

A METHOD for planning experimentation and for analyzing the data secured will be demonstrated. This method, termed *fractional replication of factorial design*, employs a relatively small number of individual experiments to determine which of a large number of process variables are controlling, to determine which combination of levels of these variables is most likely to produce optimum results, and also to predict what results will be obtained with a particular combination of conditions not already tested. Although the general method is not new, having been developed by Finney in 1945,[1] the extent to which it can improve the effectiveness of applied research generally has not been recognized by metallurgists. The fractional replication procedure is particularly useful in flotation experimentation and an example from such an investigation will be used in the paper. However, it has equal value in any investigation in which similar experimental difficulties are encountered.

In developing a flotation process for a particular mineral separation, the investigator is inevitably confronted with the following difficulties:

1—There are a large number of potentially important process variables.

2—Results of individual experiments are not highly reproducible, due in part to the difficulty in precisely controlling all the variables.

3—Considerable effort is expended in conducting individual experiments.

4—There are practical limits on the number of individual experiments which can be made.

In situations of just such a type, modern statistical methods of planning experimentation and analyzing data have their greatest value. Applications of one such technique, called *factorial design*, to problems of this type have been described by Dorenfeld and others.[2-4] The simple factorial design is an efficient procedure when the investigator hopes to provide a comprehensive understanding of the effects and interrelationships of a small number of variables over a limited range. In applied research, this is seldom the main objective. Rather, the investigator usually wishes to determine which of the many potentially important variables are in fact controlling, which levels of the controlling variables will provide optimum metallurgical results, and what these results will be at optimum conditions. Interest in detailed trends is limited to the controlling variables and to levels in the region of optimum conditions. Simple factorial design has serious deficiencies for such objectives and is not the most efficient method of experimental design.

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Deficiencies of Factorial Design

In a simple factorial design, an experiment must be made at every possible combination of each level of every variable, once these have been chosen and the levels of each to be included have been decided upon. As the number of variables or levels of each increases, the experimental program quickly reaches prohibitive size. For example, an investigation of only four variables, each at four levels, requires 256 individual experiments. Often upon completion of such an extensive program, it is found that one or more of the variables has metallurgically unimportant effects or that a poor estimate has been made as to the appropriate range of levels to be investigated. The result is that only a small proportion of the data obtained falls in the range of real metallurgical interest. Indeed, it frequently can be anticipated that certain combinations of levels of variables will not produce results of interest, but they still must be included so that the essential balance, or orthogonality, of the design will be retained. It may be true that factorial design will provide the greatest amount of information from a given number of experiments, but it does not necessarily follow that it will lead to the desired information with a minimum number of experiments. Much of the information provided may be of little value.

Advantages of Fractional Replication

The disadvantages of simple factorial design are overcome to a great extent by a modification known as *fractional replication*. This is a technique for sampling systematically the potential data of a full factorial experiment, that is, the data which would have been obtained if the complete factorially designed program had been completed. Only a fraction of the total array of experiments is made, but the experiments are chosen in such a way that the important advantages of factorial design and the accompanying analysis of variance are retained. The data obtained from the first group of experiments are used to determine which of several variables are controlling and which levels of these variables are most likely to produce the desired result. Unimportant variables and levels of variables then may be

Table I. Variables and Levels Investigated

Variable	Symbol	Level	Symbol
1—Sequence of addition of reagents	O	Frother, amine, starch	O_1
		Starch, amine, frother	O_2
2-pH	P	8	P_8
		9	P_9
3—Amount of starch	S	0.5 lb per ton	$S_{0.5}$
		1.0 lb per ton	$S_{1.0}$
		1.5 lb per ton	$S_{1.5}$
		2.0 lb per ton	$S_{2.0}$
4—Amount of amine	A	0.08 lb per ton	$A_{0.08}$
		0.16 lb per ton	$A_{0.16}$
5—Conditioning time	T	1 min	T_1
		3 min	T_3

dropped at an early stage in the program and subsequent experiments concentrated in the range of greatest interest.

Fractional Replication Illustrated

The advantages of fractional replication can be demonstrated by an example from a flotation investigation. Details of statistical theory and calculations have been omitted, since they are adequately presented elsewhere.[5-7] The general object of the investigation was to produce a very high purity iron concentrate by the flotation process. Immediate objectives were to determine: 1—which of five potentially important variables were controlling in the process, 2—which levels of these controlling variables would produce the purest concentrate, and 3—to provide enough information on the effects of variations in the levels of the controlling variables, in the vicinity of optimum conditions, to guide subsequent pilot operations. Concentrate purity was measured by silica analysis—percentage of SiO_2.

Based on the results of previous tests on similar ores and in order to settle questions raised during that experimentation, it was considered desirable to investigate the five variables listed in Table I, each variable at the levels indicated. For convenience, each variable and each level of each variable is denoted by an appropriate symbol. A full factorial design with these five variables, at the indicated levels, would require 64 individual flotation experiments. Such a program is illustrated in Table II where each item or blank represents a single experiment under the conditions indicated by the row and column headings.

Instead of 64 experiments, an initial block of only 16 tests was made, constituting a one-quarter replicate of the full factorial design. The 16 experiments appear in Table II as data figures, while the blanks are the rest of the 64 that were not conducted. The tests were run in random order using a standardized flotation test procedure, attempting as usual to keep all other conditions uniform. Tests in the initial block were chosen in a systematic manner in order to sample the potential data of the full array. General procedure for choosing the experiments has been summarized by Brownlee[6] and an arrangement similar to this one has been described by Brownlee, Kelly, and Loraine.[7]

It will suffice to point out the most important characteristic of the chosen block—its balance or orthogonality. When only these tests are considered, inspection of Table II shows that there are an equal number at each level of each variable. Thus, there are eight tests at P_8 and eight at P_9, eight at $A_{0.08}$ and eight at $A_{0.16}$, four at each level of S, etc. Furthermore, when the tests are grouped according to the levels of any chosen variable, each such group will contain an equal number of tests at each level of each other variable. For example, if all the tests at P_8 are placed in one group and all the tests at P_9 are placed in another, each group will contain four tests at O_1 and four at O_2, four tests at $A_{0.08}$ and four at $A_{0.16}$, four tests at T_1 and four at T_3, and two tests at $S_{0.5}$, $S_{1.0}$, $S_{1.5}$, and $S_{2.0}$, respectively. There are other blocks which could have been chosen which would retain this essential characteristic. The choice among blocks is dependent upon criteria to be considered later.

The data obtained from this initial block of 16 experiments reveal the relative importance of the five variables. This is not readily apparent by inspection of the data, since only in the case of a single variable O is it possible to employ the usual procedure of comparing tests that differ only in the level of that variable. But because of the balance of the design, it is possible to use analysis of variance technique to provide quantitative measures of the importance of the main effects of the variables and interactions among the variables and to provide the estimate of experimental precision necessary to judge the significance of these effects.

Analysis of variance is made in the manner usually employed for factorially designed programs; however, 16 experiments allow only 15 degrees of freedom, which must be distributed appropriately among the several main effects and interactions. The sums of squares for main effects may be calculated directly from the data using the calculation formula

$$S.S. = \frac{T_1^2 + T_2^2 + T_3^2 + \cdots\cdots T_n^2}{n} - \frac{T_{..}^2}{N}$$

where T_1, T_2, etc. are the respective sums of the data for each level of the variable, $T_{..}$ is the grand total, n is the number of tests at each level of the variable, and N is the total number of tests. Sums of squares for the two-factor interactions may be calculated in the same way from sums of appropriate groupings of the data. From the data obtained, the final analysis of variance table, shown in Table III, was developed.

With only 15 degrees of freedom available, the analysis of variance table differs from that which the full experimental program of Table II would

Table II. Full Factorial Design with Data from Experiments of Block 1, Pct SiO_2 in Concentrate

		O_1								O_2							
		P_8				P_9				P_8				P_9			
		$S_{0.5}$	$S_{1.0}$	$S_{1.5}$	$S_{2.0}$	$S_{0.5}$	$S_{1.0}$	$S_{1.5}$	$S_{2.0}$	$S_{0.5}$	$S_{1.0}$	$S_{1.5}$	$S_{2.0}$	$S_{0.5}$	$S_{1.0}$	$S_{1.5}$	$S_{2.0}$
$A_{0.08}$	T_1			0.54		0.44						0.54		0.44			
	T_3		0.54						0.52		0.52						0.52
$A_{0.16}$	T_1				0.45		0.33						0.43		0.32		
	T_3	0.32						0.39		0.31						0.37	

Table III. Analysis of Variance Block 1*

Source	Degrees of Freedom	Sums of Squares†	Mean Squares	Significance, 95 Pct Confidence
O	1	3.06	3.06	Yes
P (*SxT*, *SxA*)	1	60.06	60.06	Yes
S (*PxT*, *PxA*, *AxT*)	3	235.69	78.56	Yes
A (*PxS*, *SxT*)	1	836.55	836.55	Yes
T (*PxS*, *SxA*)	1	0.06	0.06	—
OxP	1	0.57	0.57	—
OxS	3	0.69	0.23	—
OxA	1	0.57	0.57	—
OxT	1	0.07	0.07	—
PxS (*SxT*, *SxA*)	1	18.06	18.06	Yes
Residual	1	0.56	0.56	—
Total	15	1153.94		

* Standard deviation from pooled nonsignificant interactions is 0.606 pct SiO_2.
† All data coded (x100) for ease of calculation.

have provided.* Some of the main effects and interactions have been confounded. *Confounding* is simply the statistical term for the situation which exists when tests or groups of tests being compared differ in the level of more than one of the process variables. It is not then possible to attribute any difference to a single variable, and the effects of the several variables are said to be confounded. Interactions may be confounded as well as main effects. In partial replication, the effects of several variables are intentionally confounded. In this case, however, the tests in the initial block were chosen in a particular manner so that the extent of confounding was controlled. No main effects were confounded with other main effects. Some of the main effects were partially confounded with two-factor interactions, and some of the two-factor interactions were partially confounded with other two-factor interactions, as indicated by the bracketed symbols in col. 1 of Table III. Interactions between three or more factors were assumed to be nonexistent, an assumption which is seldom proven erroneous in experimental work. That part of the variability in the data attributable to neither main effects nor two-factor interactions was lumped into the term *residual* which constitutes the preliminary estimate of experimental precision.

* The full factorial would have provided 63 degrees of freedom distributed among five main effects; 10 two-factor interactions, 10 three-factor interactions, 4 four-factor interactions, and 1 five-factor interaction.

The exact manner in which the sources of variability (main effects and interactions) are confounded is determined by the original choice of experiments in the block to be made. As mentioned previously, other orthogonal blocks of 16 could have been chosen, but the arrangement of confounding then would have been different. In order that the confounding be most economical, available knowledge of the process should be utilized in choosing the block of experiments to be made. For example, if it is known that an interaction between two particular variables is not likely to exist, it may be possible to arrange to confound that interaction with a main effect which is likely to be large. The mean square term obtained for that confounded group then may be reasonably attributed to the main effect and not to the interaction.

The fact that sources of variability are confounded does not prevent drawing statistically valid conclusions. Tests for significance can be made in the usual fashion. The two-factor interaction mean squares were tested in turn against the residual. In order to provide a better estimate of precision, the nonsignificant interaction mean squares were pooled, and the main effects were tested against this composite. Conclusions statistically correct at the indicated confidence level must be of the form "variations in *P* have a significant effect and/or an *SxT* and/or an *SxA* interaction exists."

As a practical matter, the analysis of variance provides much more than these statistically valid conclusions, although at a somewhat lower and not precisely calculable confidence level. It is evident from the relative magnitudes of the mean square terms that the amounts of amine *A*, starch *S*, and the pH *P* are probably the controlling variables in this process, decreasing in importance in the order given. Effect of differences in the sequence of addition of reagents *O*, while statistically significant, is obviously of minor metallurgical importance. Since the main effect of conditioning time *T* is not shown to be significant, it is highly unlikely that interactions involving *T* are real. Although there is independent evidence for some interactions (most probably *PxS* or *SxA*), their effect on results is likely to be minor in comparison with main effects.

The data of these 16 experiments also indicate which combination of levels of these controlling variables is most likely to produce a concentrate of minimum silica content. This information is extracted simply by averaging the data for each level of each variable, giving the data in Table IV. The averages show that the conditions most likely to produce the lowest silica content are pH 9, 0.05 lb per ton starch, 0.16 lb per ton amine, and reagent additions of starch, amine, and frother in that order. This procedure of averaging test results, no two of which were obtained under identical conditions, cannot be employed indiscriminately. Here it is valid only because of the essential balance, described previously, which is built into the experimental design. The procedure must be modified if the analysis of variance reveals important interactions.

As shown in Table II, the optimum combination of conditions had not been tested experimentally. The silica content at this condition may be estimated from the data, however, with a specified degree of precision. In this instance, it is assumed that interactions will not significantly affect the expected value. The estimate is calculated by adding a factor for each variable algebraically to the grand average. This factor for any variable is simply the average for the chosen level of that variable minus the grand average. The expected value for the conditions indicated previously is

$$\text{Expected Value} = 0.436 + (0.415 - 0.436) + (0.378 - 0.436) + (0.364 - 0.436) = 0.280 \text{ pct } SiO_2.$$

The reliability of this estimate is dependent upon

Table IV. Average Results for Each Condition

Variable	Level	Average Pct SiO_2
P	P_9	0.456
	P_5	0.415
S	$S_{0.5}$	0.378
	$S_{1.6}$	0.428
	$S_{0.5}$	0.458
	$S_{2.0}$	0.480
A	$A_{0.16}$	0.508
	$A_{0.16}$	0.364
O	O_1	0.440
	O_2	0.431
Grand average		0.436

Table V. Data of Block 2, Pct SiO_2 in Concentrate

		O_2							
		P_2				P_3			
		$S_{2.2}$	$S_{1.6}$	$S_{1.2}$	$S_{0.8}$	$S_{2.2}$	$S_{1.6}$	$S_{1.2}$	$S_{0.8}$
$A_{0.06}$	T_1	0.48		0.54		0.44		0.52	
	T_2		0.52		0.54		0.48		0.52
$A_{0.10}$	T_1		0.33		0.43		0.32		0.38
	T_2	0.31		0.40		0.27		0.37	

the precision of the data from which it was calculated (and the validity of assumption made) and can be specified by calculating confidence limits for it in the usual way. These are

$$\frac{\pm s(t_{.90,12,f})}{\sqrt{1/4}} = \frac{\pm 0.006\ (2.37)}{\sqrt{1/4}} =$$

± 0.028 pct SiO_2.

The term $\sqrt{1/4}$ derives from the fact that this is a one-quarter replicate, s is the estimated standard deviation, and t is the appropriate value from a table of student's t.* There are 19 chances out of 20 that the mean experimental value at that combination of conditions will fall within the range of 0.280 ± 0.0280 pct SiO_2. In six subsequent experiments at the indicated optimum conditions, the following silica contents were obtained: 0.27, 0.30, 0.29, 0.28, 0.28, and 0.28, averaging 0.283 pct SiO_2. Expected values for any other experimental condition may be calculated in similar fashion.

At this point, it is evident that the initial block of 16 experiments provided a wealth of information to guide subsequent experimentation. In order to further confirm that this was the optimum set of conditions, the right-hand half of the full factorial array of Table II (that part at O_2) was expanded to a one-half replicate by performing eight more experiments. These provided the data of Table V and the analysis of variance, Table VI. These data confirmed the previous conclusions in every particular. The extent of confounding was reduced by this expansion and three additional interactions were definitely eliminated as possibilities. Statistically significant evidence of an interaction (probably SxA) was obtained again, but it is evident from the relative magnitudes of the mean square terms that its effects are of minor metallurgical importance.

One new source of variability, *block difference*, was introduced. This variability is that attributable to any consistent difference between the results of the second block of experiments and those of the eight experiments from the first block. This variability arises from some unrecognized difference in conditions, such as a difference in reagent strength, temperature, etc. In this case, the block difference was not shown to be significant.

If a graph of the effects of the variables is desired, missing values may be estimated in the manner shown previously. They will be known with somewhat greater precision because of the additional data available ($\sqrt{1/4}$ in the confidence limit calculation will now be $\sqrt{1/2}$).

Any other part of the original full array of experiments could have been expanded, either to a one-half or full replicate of the factorial design. It is obvious, however, that no additional information of metallurgical importance would be obtained.

Table VI. Analysis of Variance of Block 2*

Source	Degrees of Freedom	Sums of Squares†	Mean Squares	Significance, 95 Pct Confidence
P	1	39.06	39.06	Yes
S (AxT)	3	219.19	73.06	Yes
A (SxT)	1	945.55	945.55	Yes
T (SxA)	1	0.56	0.56	—
PxS	3	1.69	0.56	—
PxA	1	0.07	0.07	—
PxT	1	0.07	0.07	—
SxA (SxT)	2	15.63	7.82	Yes
Block difference	1	1.56	1.56	—
Residual	1	3.06	3.06	—
Total	15	1226.44		

* Standard deviation from pooled nonsignificant interactions and block difference is 0.009 pct SiO_2.
† Data coded (x100) for ease of calculation.

Summary

In the example given, 24 experiments provided all of the metallurgically important information which would have required 64 experiments with simple factorial design. The first 16 experiments revealed that two of the five variables, amounts of starch and amine, were of controlling importance. They also demonstrated which set of conditions would produce optimum results and allowed reliable prediction of those results. Thus, subsequent experimentation could be concentrated in the vicinity of optimum conditions and prior commitment to an extensive experimental program could be avoided. The important advantages of factorial design and analysis of variance were retained. The effect of each variable was tested at each level of every other variable. Quantitative measures of the variability attributable to main effects, as well as those interactions most likely to exist, were provided. An estimate of experimental precision was available to test the significance of those effects and to judge the reliability of conclusions. Enough data were made available so the trends of the effects of the several variables could be judged. In the type of problem most frequently encountered in applied research and development, this modification of the factorial design procedure greatly improves its efficiency and provides the desired information with minimum experimental effort.

Appendix

For the benefit of those technically interested, the experimental design described herein is based on a 2^5 factorial. The factor at four levels is regarded as two pseudo-factors S' and S'', each at two levels. The alias subgroup is $I = OPS'S''AT$. The three interactions confounded to generate the four blocks are $PS'T$, $PS''A$, and $S'S''AT$. Each source then has three aliases as follows:

O	$= OPS'T$	$= OPS''A$	$= OS'S''AT$
P	$= S'T$	$= S''A$	$= PS'S''AT$
S'	$= PT$	$= PS'S''A$	$= S''AT$
S''	$= PS'S''T$	$= PA$	$= S'AT$
$S'S''$	$= PS''T$	$= PS'A$	$= AT$
A	$= APS'T$	$= PS''$	$= S'S''T$
T	$= PS'$	$= PS''AT$	$= S'S''A$
OP	$= OS'T$	$= OS''A$	$= OPS'S''AT$
OS'	$= OPT$	$= OPS'S''A$	$= OS''AT$
OS''	$= OPS'S''T$	$= OPA$	$= OS'AT$
$OS'S''$	$= OPS''T$	$= OPS'A$	$= OAT$
OA	$= OPS'AT$	$= OPS''$	$= OS'S''T$
OT	$= OPS'$	$= OPS''AT$	$= OS'S''A$
$PS'S''$	$= S''T$	$= S'A$	$= PAT$
$OPS'S''$	$= OS''T$	$= OS'A$	$= OPAT$

Acknowledgments

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Discussion of this paper, if any, will appear in JOURNAL OF METALS, November 1956, and in AIME Metals Branch *Transactions*, Vol. **206**, 1956.

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Technical Note

Simplification of a Molten Zone Refining Formula

by Keith S. Milliken

NORMAN W. Lord[1] has shown that the resultant impurity distribution after n zone passes is given by

$$C_n(a) = C_0\left\{1-(1-k)e^{-ka}\left[n-\sum_{s=1}^{n-1}\sum_{r=1}^{s}k^{s-r}e^{-ka}f_r^{(s)}a^r\,(r+1-ka)\right]\right\}$$

in the ingot section $0 \leq a < N-n$, where a is the displacement in zone lengths, N is the ingot length in zone lengths, C_0 is the initial uniform impurity concentration, k is the segregation constant $= C_{solid}/C_{liquid}$ at equilibrium, and

$$f_r^{(s)} = \frac{s^{s-r-1}}{(s-r-1)!\,r!}\,.$$

The foregoing expression is formidable in respect to computational labor. However, the triple sum can be reduced as follows

$$\sum_{r=0}^{s-1} f_r^{(s)}\,a^r = \frac{s^{s-1}}{(s-1)!}\sum_{r=0}^{s-1}\frac{(s-1)!}{(s-r-1)!\,r!}\left(\frac{a}{s}\right)^r = \frac{(s+a)^{s-1}}{s!}$$

using the binomial theorem. Similarly

$$\sum_{r=0}^{s-1} f_r^{(s)}\,ra^r = a(s-1)\,\frac{(s+a)^{s-2}}{s!}$$

giving

$$\sum_{r=0}^{s-1} f_r^{(s)}\,a^r\,(r+1-ka) = \frac{(s+a)^{s-2}}{s!}\,\{a(s-1)+(1-ka)(s+a)\}\,.$$

Finally

$$\frac{C_n(a)}{C_0} = 1-(1-k)e^{-ka}\left[n-\sum_{s=1}^{n-1}\sum_{r=1}^{s}k^{s-r}e^{-ka}\frac{(s+a)^{s-1}}{s!}\{a(s-1)+(1-ka)(s+a)\}\right]$$

may be used if $C_n(a)$ is required for a series of values of n, and

$$\frac{C_n(a)}{C_0} = 1-(1-k)e^{-ka}\left[n-\sum_{s=1}^{n-1}(n-s)k^{s-1}e^{-ka}\frac{(s+a)^{s-2}}{s!}\{a(s-1)+(1-ka)(s+a)\}\right]$$

is better if computing for only a single value of n.

For $k<1$, it can be shown that this last equation is equivalent to

$$\frac{C_n(a)}{C_0} = \frac{1-k}{k}\,e^{-ka}\sum_{s=n+1}^{\infty}(s-n)\frac{(s+a)^{s-2}}{s!}\,(ke^{-a})^s\,\{a(s-1)+(1-ka)(s+a)\}\,.$$

The infinite series is useful only when k is small enough to force rapid convergence. In such cases, the absence of subtraction from unity has the definite advantage of preserving accuracy.

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TN 268E. Manuscript, Nov. 16, 1954.

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Diffusion in Liquid Lead-Bismuth Alloys

by R. E. Grace and G. Derge

Diffusivity of bismuth in liquid Pb-Bi alloys has been measured by the capillary reservoir method as a function of temperature and composition. Fair agreement between theory and experiment is found for the measured diffusion coefficients and activation energies of diffusivity and viscosity in dilute lead alloys. The measured diffusion coefficients in the high bismuth alloys describe mass transport in special concentration gradients.

THE study of diffusion in liquid metals has been restricted largely to the amalgams over small ranges of temperature and composition. In contrast to this, diffusion and transport of ions and molecules in aqueous and organic solvents have been studied extensively for the past 50 years. The general difficulty of measuring diffusion coefficients in both gases and liquids at high temperatures has been that of eliminating convection currents. Measuring techniques based on freezing and sectioning after the diffusion period[1] are not particularly satisfactory because of the volume changes and segregation which occur during freezing.

The most promising technique which is currently being developed for liquid metals is the capillary reservoir method. This eliminates convection currents from the diffusion zone and requires measurements only of the length and initial and final compositions of the alloy within the capillary. Anderson and Saddington[2] have measured the diffusion coefficient of tungstate anions in aqueous Na_2WO_4; Wang[3, 4] and Hoffman[5] have studied structure and self-diffusion in water and self-diffusion in mercury, respectively, by similar techniques.

Selection of the Pb-Bi system for initial study was made for several reasons: 1—It afforded a low temperature system for development of techniques which would be suitable for high temperature measurements in liquid iron. This research on diffusion in liquid iron is now in progress as a part of a general program on the rates and mechanisms of slag-metal reactions. 2—Activity data on liquid Pb-Bi alloys of all compositions are available from the *emf* measurements of Strickler and Seltz.[6] 3—Some data are available on the diffusion of bismuth into solid lead. The measurements on solid alloys were made by von Hevesy and Seith[7] for the concentration range of 0 to 2 pct Bi.

An appropriate solution to Fick's law (unsteady state) for the capillary reservoir has been found by considering the mass flow which passes through a unit area of surface on a slab of infinite thickness as a function of time. This is the first application of this solution to diffusion measurements, although a similar solution is available for identical boundary conditions.[8] By analogy with heat conduction, the problem reduces to a perfectly insulated infinite slab at temperature C_o, one surface of which is quenched to C_s for all times, t. Heat is permitted to flow from quenched surface of the slab to an infinite supply of coolant at C_s.* (see Appendix).

For the corresponding solution for diffusion in liquids, these terms become

$$\frac{C_o - C_o}{C_s - C_s} = \frac{2}{l}\left[\frac{Dt}{\pi}\right]^{1/2} \qquad [1]$$

where C_o is the initial concentration of component A* in entire capillary at $t = 0$, C_o is the final concentration of component A in entire capillary at $t = t$, C_s is the concentration of component A in bath for all t, l is the length of the capillary in centimeters, t is the time of diffusion period in seconds, and D is the diffusion coefficient in square centimeters per second.

By comparison with solutions to Fick's law used in the solid state, this solution for liquids is similar to the Grube solution for solids, where the change in D with chemical composition is not taken into consideration. Difficulty is to be expected if large concentration gradients are employed and if the concentration gradients extend over an appreciable length of the capillary.

* In this case, component A is bismuth, and $C_s = 0$.

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Discussion of this paper, TP 3979D, may be sent, 2 copies, to AIME by Sept. 1, 1955. Manuscript, Feb. 8, 1954. New York Meeting, February 1954.

Table I. Data for All Diffusion Runs with Unstirred Bath*

Run No.	Time, Min	l, Cm	C_x, Pct Bi	°C	D_{Pb}^{Pb} x10⁵, Sq Cm per Sec
275a	1110	5.68	1.15	345	3.2
275b	1110	4.80	1.09	345	3.7
276a	750	3.66	0.99	345	4.1
277a	2†	3.20	1.70	245	—
278a	875	4.70	1.10	431	4.1
278b	875	3.63	0.90	431	4.4
279a	1200	4.68	0.89	431	5.4
279b	1200	4.62	1.01	431	4.2
280a	2†	3.58	1.75	431	—
280b	2†	3.63	1.68	431	—
282b	990	3.28	0.60	518	6.0
283a	785	4.17	0.91	518	6.2
284a	360	5.72	1.30	518	6.6
284b	360	7.45	1.43	518	5.1

* C_0 = 1.70 pct Bi and C_x = 0.00 pct Bi. Capillary of 1.20 mm bore.
† Blank runs.

The test of whether the capillary is truly infinite is made by running check experiments with different diffusion times or capillary lengths. If the diffusion coefficients agree in these cases, the boundary conditions probably have been fulfilled.

Viscosity and diffusivity are closely related rate processes and their temperature dependence is very similar.[5] The Einstein-Stokes and the Eyring equations on relating these terms have identical temperature coefficients, but the diffusivities differ by a factor of about 3π in magnitude. The Einstein-Stokes relation

$$D = \frac{kT}{6\pi\eta r} \qquad [2]$$

is based on the medium being continuous. The Eyring equation takes the discontinuity in "jump-distances," $\lambda_x = \lambda_y = \lambda_z = 2r$, into account, where x,y, and z refer to the coordinates of the system and r is the atomic radius for the case of simple monatomic liquids. The Eyring equation is

$$D = \frac{\lambda_o\, kT}{\lambda_x\lambda_y\eta}. \qquad [3]$$

In both Eqs. 2 and 3, the exponential dependence of η, the coefficient of viscosity, on T governs the relation between D and T.

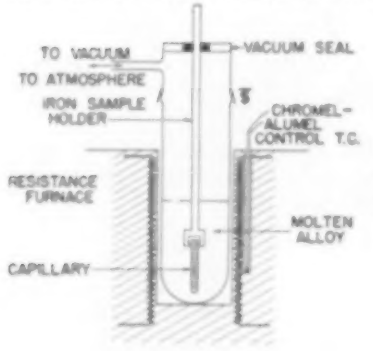

Fig. 1—Diagram shows the apparatus used to fill the capillaries.

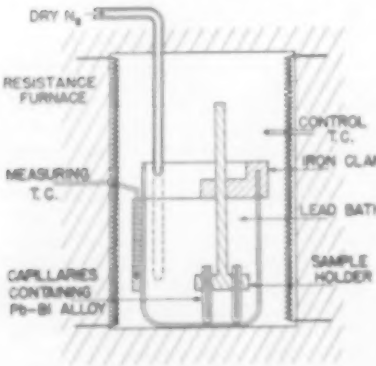

Fig. 2—The diffusion apparatus is illustrated in the diagram. N_2 bubbler can be lowered into the bath for stirring.

Experimental Technique

These measurements were made with alloys of high purity lead and bismuth (99.99 pct) melted in Pyrex flasks under vacuum. Initial concentrations of 1.7, 52.7, and 100 pct Bi were diffused from the capillaries into a 16 kg bath of liquid lead. The 1.7 pct Bi alloys were run first; and after the final 100 pct Bi runs were made, the composition of the lead bath was greater than 99.8 pct Pb, with bismuth being the major impurity. The only other sources of impurity additions were stirring rods and sample holders. In all cases these were made of Pyrex or low carbon iron. The solubility of iron in lead and in bismuth is less than 10^{-4} pct.[6]

Pyrex capillaries of 0.79 and 1.20 mm bore and 3 to 9 cm in length were sealed at one end. A clean fractured surface normal to the cylinder axis insured a uniform cross-section at the capillary throat. The constancy of the bore sizes was checked by measurement at 20X. These capillaries were filled by immersing them in a given alloy under vacuum and then opening the system slowly to the atmosphere in order to force the liquid alloy into the capillary. The filled capillaries were allowed to cool in air and were stored in a desiccator until used. The filling apparatus is shown in Fig. 1.

The diffusion runs were made by submerging the capillaries in a constant temperature lead bath, controlled to ±2°C in a totally enclosed resistance furnace. This set-up is shown schematically in Fig. 2. A temperature variation of 6°C was found in the 4 in. bath depth by exploration with a chromel-alumel thermocouple. The bath was hotter at the bottom. The resulting convection was the only stirring provided in most of these experiments, and they are classified as "unstirred." Generally, two capillaries were run at the same time, but frequently one would break due to thermal shock. Missing run numbers in the tables (I to III) are for those capillaries which broke, or in which a gas bubble formed, or where the diffusion time was too long for the boundary conditions to apply. Only single samples were dif-

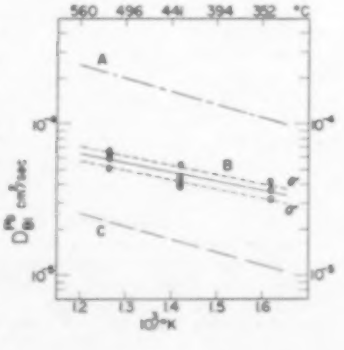

Fig. 3—Data for the diffusivity of bismuth into liquid lead is illustrated. Line A is the Eyring calculation of diffusivity of bismuth into lead at infinite dilution. Lines B are the measured diffusivity of bismuth into lead when C_0 = 1.7 pct Bi and C_x = 0.00 pct Bi. Capillary of 1.20 mm bore and unstirred bath. Line C is the Einstein-Stokes calculation of diffusivity of bismuth into lead at infinite dilution.

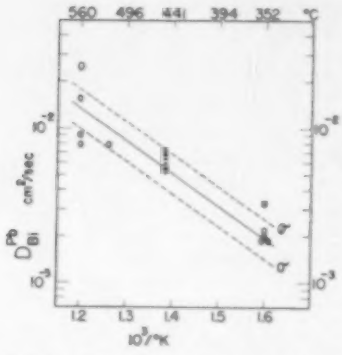

Fig. 4—Diffusivity of bismuth into lead is illustrated when C_o = 52.7 pct Bi. Capillary of 1.20 mm bore and unstirred bath.

fused in the 52.7 pct Bi runs, because the capillaries had to be handled quickly due to the low melting temperature of the alloy.

Discussion

Fig. 3 shows the data for the diffusivity of bismuth into liquid lead when an initial concentration of 1.7 pct Bi was used. The scatter in the data is partly due to uncertainties in the chemical analyses after the diffusion runs. The analyses are good to ±0.05 pct Bi in the range below 5 pct and are good to 0.5 pct Bi in the higher alloys. The calculation of diffusivity by either the Einstein-Stokes or the Eyring equations is not in agreement with the data. A value of 1.46A was used as the effective radius of the bismuth atom,[11] and values of viscosity of liquid lead were taken from Bienias and Sauerwald.[12] The energy of activation of viscosity, ΔH°_ν, of pure liquid lead is 2380 cal per mol; for the 1.7 pct Bi alloy, it is 2040 cal per mol. The line of best fit for the diffusivity data, calculated by the least squares method, is

$$D_{Bi}^{Pb} = 3.37\text{x}10^{-4} \exp\left[-\frac{2760}{RT}\right] \text{ sq cm per sec} \quad [4]$$

and C_o = 1.7 pct Bi. Thus, from the measured diffusivities, ΔH°_D = 2760 cal per mol which compares with ΔH°_ν = 2040 to 2380 cal per mol for viscosity.[13]

In comparing these liquid data with values in the literature for solid diffusion in 1 to 2 pct Bi-Pb alloys,[7] the diffusivity increases at the melting temperature by a factor of about 10^4. A single measurement of the self-diffusion of liquid lead has been made by Groh and von Hevesy.[13] Their value of $2.54\text{x}10^{-4}$ sq cm per sec at 343°C is in fair agreement with the $3.5\text{x}10^{-4}$ sq cm per sec measured in the 1.7 pct Bi alloy.

Table II. Data for All Diffusion Runs with Unstirred Bath*

Run No.	Time, Min	l, Cm	C_o, Pct Bi	°C	Apparent D_{Bi} x10^5, Sq Cm per Sec
150	13	6.20	39.9	356	2.2
151	20	6.56	34.6	356	3.3
152	10	6.20	42.2	356	2.0
154	8	5.39	42.2	356	1.9
155	8	5.52	42.4	356	1.9
156	9	7.32	38.5	450	5.7
158	7	9.35	41.7	450	7.2
159	10	9.10	40.8	450	5.5
160	15	8.89	36.3	450	6.6
163	20	5.21	17.7	520	7.6
164	9	8.95	33.3	560	16.0
165	5	6.84	39.6	560	7.9
166	4	6.00	45.1	560	25.0
167	6	7.09	40.7	560	9.1

* C_o = 52.7 pct Bi and C_x = 0.00 pct Bi. Capillary of 1.20 mm bore.

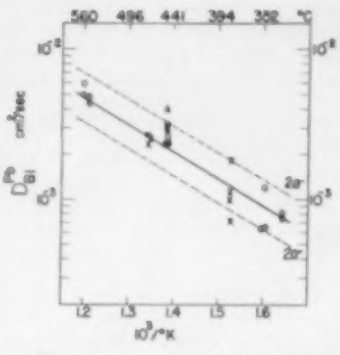

Fig. 5—Diffusivity of bismuth into lead is shown when C_o = 100 pct Bi. Open circle represents capillary of 1.20 mm bore and unstirred bath; X, 0.79 mm bore and unstirred bath; triangle, 1.20 mm bore and bath stirred at two bubbles per second; and closed circle, 1.20 mm bore and bath stirred at four bubbles per second.

Figs. 4 and 5 show the data for diffusivity of bismuth into liquid lead when the initial compositions in the capillaries are 52.7 and 100 pct Bi. The least squares lines for the 1.20 mm bore, unstirred runs are

$$\text{Apparent } D_{Bi} = 4.1 \exp\left[-\frac{9500}{RT}\right] \text{ sq cm per sec} \quad [5]$$

where C_o = 52.7 pct Bi, ΔH°_D = 9500 cal per mol compared with ΔH°_ν = 1460 cal per mol,[12] and

$$\text{Apparent } D_{Bi} = 6.5 \exp\left[-\frac{8150}{RT}\right] \text{ sq cm per sec} \quad [6]$$

where C_o = 100 pct Bi and ΔH°_D = 8150 cal per mol compared with ΔH°_ν = 1440 cal per mol.[14]

The tests for effects of bore size and stirring rate are shown in Fig. 5, for the 100 pct Bi runs. This graph shows the two σ limits for the unstirred runs with normal capillary size. Neither increased bore size nor stirring rate appears to have significant importance in the measured diffusion rates. The boundary condition at the open throats of the capillaries is maintained by the very small convective stirring present in the diffusion bath during the run.

In Table I the diffusion coefficients reported are D_{Bi}^{Pb}, the true diffusion coefficient of bismuth in a Pb-Bi alloy of 0 to 2 pct Bi. In Tables II and III the apparent diffusivities of bismuth in the various con-

Table III. Data for All Diffusion Runs*

Run No.	Bore, Mm	Time, Min	l, Cm	C_o, Pct Bi	Stirring	°C	Apparent D_{Bi}, Sq Cm per Sec
102b	1.20	90	6.00	64.4	No	354	$6.4\text{x}10^{-6}$
103b	1.20	10	5.34	88.8	No	351	$6.5\text{x}10^{-6}$
105a	1.20	6	5.85	87.5	No	351	$1.2\text{x}10^{-5}$
109a	1.20	4	4.88	89.9	No	336	$7.9\text{x}10^{-6}$
110a	1.20	8	4.57	85.0	No	336	$7.7\text{x}10^{-6}$
112a	1.20	15	3.96	40.8	No	562	$4.8\text{x}10^{-5}$
113a	1.20	15	4.12	38.8	No	562	$5.9\text{x}10^{-5}$
114b	0.79	21	3.46	51.5	No	392	$1.8\text{x}10^{-5}$
115b	0.79	40	4.12	56.2	No	382	$1.1\text{x}10^{-5}$
118a	0.79	5	3.86	83.9	No	382	$1.0\text{x}10^{-5}$
118b	0.79	5	4.37	88.1	No	382	$7.1\text{x}10^{-6}$
119a	0.79	20	3.51	41.8	No	473	$2.7\text{x}10^{-5}$
119b	0.79	20	3.61	48.5	No	473	$2.3\text{x}10^{-5}$
123b	0.79	7.5	2.80	56.2	No	470	$2.6\text{x}10^{-5}$
125b	1.20	15	4.57	62.5	No	449	$2.6\text{x}10^{-5}$
127b	1.20	20	4.75	59.0	A†	449	$2.5\text{x}10^{-5}$
128b	1.20	35	5.24	37.1	A	449	$4.1\text{x}10^{-5}$
129a	1.20	5	3.86	75.4	B†	449	$2.4\text{x}10^{-5}$
132b	1.20	15	5.01	55.3	No	592	$4.4\text{x}10^{-5}$
135a	1.20	6	6.54	77.3	No	552	$4.8\text{x}10^{-5}$
136a	1.20	15	5.54	67.1	B	450	$2.9\text{x}10^{-5}$
137a	1.20	15	5.18	64.0	B	450	$3.0\text{x}10^{-5}$

* C_o = 100 pct Bi and C_x = 0.00 pct Bi.
† Rate A is two bubbles per second and B is four bubbles per second.

centration intervals are shown. These diffusivities were calculated by using Eq. 1. The limits of certainty of these data can be determined by reference to Figs. 3, 4, and 5. In a future paper, the nature of the concentration gradient and its effect upon the true diffusion coefficient of bismuth in various Pb-Bi alloys will be reported.

Summary

1—Diffusivity of bismuth in liquid Pb-Bi alloys has been measured as a function of initial concentration and temperature by the capillary reservoir technique.

2—The Einstein-Stokes and the Eyring equations are not adequate to represent the diffusion coefficient of bismuth in dilute solution in liquid lead. It is not possible to evaluate the relative merit of the two equations.

3—The activation energies of diffusion and viscosity are approximately the same in the dilute alloys; however, ΔH°_ν is much greater than ΔH°_ν in the higher bismuth alloys when a large concentration gradient is employed.

Appendix: Solution of Fick's Law for Long Capillary

Fick's second law for unidirectional diffusion may be written

$$\frac{\partial C}{\partial t} = D\,\frac{\partial^2 C}{\partial x^2} \quad * \qquad [7]$$

* Nomenclature for the equations is as follows: C is concentration, C_i is the initial concentration, C_s is the surface concentration, C_a is the average concentration in the capillary at time t, D is the diffusion coefficient, l is the length of the capillary, A is the cross-sectional area of the capillary, t is the time of diffusion, q_s is the rate of diffusion of the solute from the open end of the capillary, and Q_s is the quantity of solute diffused from the capillary in time t.

assuming the variation of diffusion coefficient with concentration can be neglected. The boundary conditions to be considered are as follows:

1—C ($x = x$ and $t = 0$) $= C_i$. At the start, the concentration in the capillary is uniform and equal to C_i.

2—C ($x = 0$ and $t = t$) $= C_s$. The surface concentration at the open end of the capillary is suddenly changed to C_s and maintained constant at C_s throughout the experiment.

3—C ($x = \infty$ and $t = t$) $= C_i$. The capillary is considered to be of infinite length; practically speaking, the concentration changes resulting from diffusion should not penetrate the full length of the capillary.

For these boundary conditions, the solution to Eq. 7 may be written by analogy with the corresponding solution for unsteady heat flow in an infinitely thick plane wall[9]

$$C = C_s + (C_i - C_s)\left\{1 - erf\left(\frac{x}{2\sqrt{Dt}}\right)\right\} \qquad [8]$$

in which *erf* refers to Gauss's error integral.

Fick's first law gives the rate of diffusion q_s out the open end of the capillary (at $x = 0$)

$$q_s = DA\left(\frac{\partial C}{\partial x}\right)_{x=0} \cdot \qquad [9]$$

The concentration gradient at the open end is obtained by first differentiating Eq. 8 with respect to x and then substituting $x = 0$ in the result. Combining this result with Eq. 9

$$q_s = A\,(C_s - C_i)\left(\frac{D}{\pi t}\right)^{1/2} \cdot \qquad [10]$$

The total quantity of solute Q_s diffused out of the capillary in time t is given by

$$Q_s = \int_0^t q_s\,dt\,. \qquad [11]$$

Substituting Eq. 10 in Eq. 11 and integrating

$$Q_s = 2A\,(C_s - C_i)\left(\frac{Dt}{\pi}\right)^{1/2} \cdot \qquad [12]$$

The original quantity of solute in the capillary is C_iAl and the final quantity is C_aAl. Thus, the materials balance for the process can be written

$$C_iAl = C_aAl + 2A\,(C_s - C_i)\left(\frac{Dt}{\pi}\right)^{1/2} \cdot \qquad [13]$$

Rearrangement of Eq. 13 gives

$$\frac{C_s - C_a}{C_s - C_i} = \frac{2}{l}\left(\frac{Dt}{\pi}\right)^{1/2} \qquad [14]$$

which is the desired solution.

In the foregoing derivation, it is implicitly assumed that concentrations are in quantity of solute per unit volume of solution and that there is no superimposed mass flow out the end of the capillary as the result of changes in molar volume within the capillary. If density changes occur and if concentrations are expressed in weight percent, Eq. 14 must be regarded as an approximation.

Acknowledgments

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Discussion of this paper, if any, will appear in JOURNAL OF METALS, November 1956, and in AIME Metals Branch *Transactions*, Vol. **206**, 1956.

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AIME OFFICERS:
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aime NEWS

Metals Branch Secretary Appointed

The new man behind the work-laden desk of the Metals Branch Secretary is Robert William Shearman. Mr. Shearman thus assumes the role left vacant by the promotion of Ernest Kirkendall to Secretary of the Institute.

The son of a mining engineer, Mr. Shearman is a native of Palmerton, Pa., and a graduate of the Lawrenceville School and Princeton University. He graduated from the latter in 1937 with a bachelor's degree in chemistry. Graduate courses at Polytechnic Institute of Brooklyn, Pennsylvania State College (now University) Extension, Harvard University, Columbia University, and New York Institute of Finance filled in additional background in economics, metallurgy, and chemistry.

Before joining the AIME staff, Mr. Shearman was with Opinion Builders, Inc., New York City public relations firm, where he specialized in technical publicity.

Following two years with Pittsburgh Plate Glass Co., he was employed by Bethlehem Steel Co. as a metallurgical observer, working in the open hearth and electric furnace shops. From 1946 to 1951 Mr. Shearman was with Union Carbide & Carbon Corp. He created advertising and publicity material for the alloys and metals div., working on products of the Electro Metallurgical Co., and Haynes Stellite Co. He also worked on the corporation's annual report. Mr. Shearman later went to the industrial gases div., working on process and products publicity for the Linde Air Products Co., covering oxygen in steel production, jet-piercing of taconite ores, welding equipment and processes, and powder cutting and scarfing. He assisted in the work of the program, publicity, and arrangements committee of the International Acetylene Assn. during that period.

In 1951 he was recalled to the U. S. Air Force, but returned to Union Carbide in 1953 to become supervisor and later assistant division manager of the general activities div., general publicity dept.

Mr. Shearman first entered the armed services as a Medical Corps private in 1941, rising to sergeant. After attending officer candidate school at Edgewood Arsenal, Mr. Shearman stayed on as a Chemical Warfare OCS instructor. Later he entered the Air Force and trained at Air Force Weather School, Air Force Intelligence School, Air Force School of Applied Tactics, and Command and General Staff School. As a combat intelligence officer in the Pacific Theater, he went from lieutenant to captain. It was as a member of the Air Force Reserve that he was recalled to duty in 1951. By that time he had risen to major, a rank he now holds.

ROBERT W. SHEARMAN

One of Mr. Shearman's abiding interests is hiking. He hopes someday to cover the entire Appalachian Trail from Maine to Georgia. He sees it as a plan for some time in the dim future. Mr. Shearman has hiked in the Green and White Mountain Ranges of New England, tramped through Colorado, and has done considerable walking in Europe and Canada. He is a member of the Appalachian Mountain Club. In addition to European travels, Mr. Shearman has been to Mexico and has several Canadian trips to his credit.

A number of technical and non-technical articles over his byline have appeared in newspapers, magazines, and journals. Married in 1954 to the former Dorothy French Dodge of Mountain Lakes, N. J., Mr. Shearman makes his home at 2 Stuyvesant Oval, New York City.

Division Officers Nominated for 1956

Iron and Steel Div.

Chairman: John Chipman, MIT; *Vice Chairmen:* H. B. Emerick, Jones & Laughlin Steel Corp.; John Golden, U. S. Steel Corp.; Kurt Neustaetter, Inland Steel Co.; *Secretary-Treasurer,* Robert W. Shearman, AIME. *Executive Committee* (1956 to 1959): H. M. Kraner, Bethlehem Steel Co.; R. P. Liggett, Republic Steel Corp.; W. E. Marshall, Armco Steel Corp.

Extractive Metallurgy Div.

Chairman (Automatic): J. C. Kinnear, Jr., Kennecott Copper Corp.; *Chairman-Elect:* Bruce Gonser, Battelle Memorial Institute; *Secretary:* Robert W. Shearman, AIME. *Executive Committee* (1956 to 1959): N. C. Fick, Dept. of Defense; T. K. Graham, Anaconda Co.

Institute of Metals Div.

Chairman: C. S. Barrett, University of Chicago; *Senior Vice-Chairman:* W. R. Hibbard, Jr., General Electric Co.; *Vice Chairman:* W. J. Harris, Jr., Battelle Memorial Institute; *Secretary-Treasurer:* Robert W. Shearman, AIME. *Executive Committee* (1956 to 1959): Amos J. Shaler, Pennsylvania State University; J. H. Jackson, Battelle Memorial Institute; E. S. Rowland, Timken Roller Bearing Co.

The AIME Board of Directors has agreed to a proposal that a further study be made of the best location for a new engineering building, the Pittsburgh offer meanwhile being held in abeyance. A full report will appear in the August issue.

1956 AIME Nominations Slate Announced

For President-Elect

GROVER J. HOLT

Grover J. Holt, general manager of the ore mining dept., Cleveland-Cliffs Iron Co., Ishpeming, Mich., has been nominated for the presidency of AIME in 1957. Mr. Holt was born in Northwood, N. D. on July 30, 1894. He enrolled in mining engineering at the University of North Dakota, received a bachelor of science in mining engineering in 1916 and later received the degree of mining engineer from that university. Mr. Holt was engaged in postgraduate work at the University of Utah from 1916 to 1917. This work was in connection with a research fellowship in metallurgy and ore dressing. He spent a short time following completion of this fellowship work as superintendent of the Basin Metal Mining Corp., Ouray, Utah. During World War I, Mr. Holt entered the U. S. Army Engineer Corps at Corpus Christi, Texas, entered officers' training, and served in France.

During 1919 he was employed by the Bunker Hill & Sullivan Mining & Smelting Co., Kellogg, Idaho. In 1920 he joined the U. S. Steel Corp. at Chisholm, Minn., remaining as a mining engineer until 1928. For one year he was chief engineer and superintendent of the Manganiferous Iron Co. of Crosby, Minn. From 1930 to 1942 he was located at the St. Paul office of Butler Brothers as assistant to the vice president.

Mr. Holt joined Cleveland-Cliffs in 1942 as chief engineer. He was promoted to manager of Minnesota mines in 1950 and was appointed general manager of mines for Minnesota and Michigan in 1953.

Mr. Holt has been extremely active in AIME and aided in the formation of the Minerals Beneficiation Div. He was Chairman of that division, and also Chairman of the Minnesota Local Section and the annual fall meeting of the MBD held in Hibbing in 1953. He served as a member of the Nominating Committee and was appointed Chairman of that committee in 1953.

For Vice Presidents

L. E. ELKINS

Lloyd E. Elkins, production research director, Stanolind Oil & Gas Co., has been nominated as Vice President and Director of AIME. Born at Golden, Colo., in 1912, he received his degree in petroleum production engineering from the Colorado School of Mines. Following graduation he was employed by the Stanolind Oil & Gas Co. at Wink, Texas, and has risen steadily during his years with the company. In 1948 Mr. Elkins was selected to attend the advanced management program at the Harvard Graduate School of Business Administration. He has been active in the Petroleum Branch of AIME of which he was Chairman in 1949.

J. L. GILLSON

Joseph Lincoln Gillson became an AIME Member in 1923. He is now nominated as a Vice President. He received his B.S. and M.A. degrees from Northwestern and his M.S. from MIT, where he was an associate professor in 1928. The following year he returned to Northwestern, as associate professor of economic geology. In 1930 he joined E. I. du Pont deNemours & Co., Wilmington, Del. Mr. Gillson was Chairman of the Industrial Minerals Div., AIME in 1947 and was also Chairman of the AIME Committee on Democratization. He was a Vice President of AIME 1951-54.

For Directors

R. B. CAPLES

R. B. Caples, has been nominated as a Director of AIME. He was born in Glasgow, Mo., in 1888 and graduated from the Missouri School of Mines in 1919, where he received an honorary degree of Doctor of Engineering in 1948. Mr. Caples started his professional career with the Federal Mining & Smelting Co. in the

Edward G. Fox, Chairman of the Nominating Committee, has announced nominations for President-Elect, Vice Presidents, and Directors for 1956. As provided in Art. IX, Sec. 2 of the bylaws, 25 Members or Associate Members may sign and transmit to the Secretary's office prior to September 1, "any complete or partial ticket of nominees," should they wish other candidates to be considered.

In such instance, a letter ballot will be forwarded to all Members in good standing in the United States, Canada, and Mexico, tabulating both the official ballot and any supplementary nominations. If no supplementary nominations are thus received, no letter ballot will be printed, and nominees on the official ballot shall be declared duly elected at the meeting of the Board of Directors or the Executive Committee in November.

Coeur d'Alene district in Idaho and shortly thereafter entered the employ of the Anaconda Copper Mining Co. He was first employed in the testing dept. of the Anaconda Reduction Works, working in collaboration with Messrs. Frederick Laist, F. F. Frick, and James O. Elton in the development of the electrolytic zinc process during 1924 to 1916, as well as the basic process for the Anaconda phosphate plant. In 1918 he became superintendent of the Great Falls, Mont., plant and was made manager in 1941. Mr. Caples is presently president and director of the Anaconda Aluminum Co., New York, N. Y. He has served on various committees of the Extractive Metallurgy Div. of the Institute, including the Executive Committee and has just completed a three year term as AIME Director.

C. R. DODSON

Charles R. Dodson has been nominated as Director for a three year term. Mr. Dodson is assistant vice president at the First National City Bank of New York, New York, N. Y. He was born in Washington, D. C. in 1907 and graduated from University of Maryland and Massachusetts Institute of Technology. He was at the University of California from 1932 to 1934, engaged in graduate work in mathematics, physics, and engineering. In 1944 Mr. Dodson enrolled at the University of Southern California for a course in supervision and management. During his career he has been associated with the General Electric Co., York Ice Machinery, and Standard Oil Co. of California. Mr. Dodson has been active in the petroleum industry for 19 years. From 1947 to 1954 he has been engaged in consulting and teaching. In March 1954, Mr. Dodson joined the petroleum dept. of the First National City Bank of New York.

ANDREW FLETCHER

Andrew Fletcher, president of St. Joseph Lead Co., has been nominated as a Director of AIME for three years. Mr. Fletcher, who was born in 1895 in New York City, was educated at Hill School and the Sheffield Scientific School of Yale University, where he received a Ph.B. in mechanical engineering. In 1949 the University of Missouri conferred upon him an honorary degree of Doctor of Engineering. In addition to being president of St. Joseph Lead Co., he is also president of the Mine LaMotte Corp., and Bonne Terre Farming & Cattle Co. He is vice-president of Missouri-Illinois R. R. Co. and director of the America-Hawaiian Steamship Co. He has been on the Board of AIME for many years and has been Treasurer and President.

C. R. KUZELL

C. R. Kuzell, vice president in charge of western activities for the Phelps Dodge Corp., Douglas, Ariz., has been nominated as an AIME Director for a three-year term. Mr. Kuzell is also president of the Apache Powder Co., Benson, Ariz. He is a graduate of the Case School of Applied Science. In 1910 Mr. Kuzell joined the Anaconda Copper Mining Co., Great Falls, Mont. He was transferred to Anaconda as superintendent of reverberatory smelting in 1914. From 1918 to 1935 Mr. Kuzell was associated with the United Verde Copper Co., Clarkdale, Ariz., where he was smelter superintendent and later became general superintendent. United Verde became a branch of Phelps Dodge Corp., and Mr. Kuzell was manager from January 1940 to 1944. He was then appointed director of labor relations and consulting metallurgist for Phelps Dodge at Douglas. He was named assistant general manager in 1946 and held this position until 1952 when he became general manager. Mr. Kuzell has held his present position since January 1955. Mr. Kuzell holds membership in the American Mining Congress, American Chemical Society, and Mining and Metallurgical Society of America. He has been an AIME Member since 1912. The Arizona Section named him Chairman in 1946 and he was also Chairman of the Committee on Reduction and Refining of Copper in 1933.

F. J. MEEK

Fred J. Meek, plant engineer for the American Zinc Co. of Illinois, E. St. Louis, Ill., has been nominated for a three year term as Director of AIME. Mr. Meek has served the St. Louis Local Section as Secretary, Chairman, Council Delegate, and Secretary for the 1951 Annual Meeting Committee. He was Chairman of the Council of Section Delegates in 1954 and served on the National Membership Steering Committee and Nominating Committee in 1954. Mr. Meek attended the University of Illinois taking mechanical engineering and later mining engineering courses. His schooling was interrupted by World War I. He entered the Air Force in 1917 and received his commission as a second lieutenant. He returned to mining work with the Egyptian Coal & Mining Co. and O.K. Coal Co. In 1932 Mr. Meek joined the Eagle-Picher Lead Co., Hillsboro, Ill., later transferring to the mining and smelting div. He became plant engineer for the American Zinc Co. of Illinois in 1941 at E. St. Louis, Ill. Continuing in that position he was put in charge of engineering and later became superintendent of the power and mechanical depts.

G. F. MOULTON

Gail F. Moulton, presently a Director of AIME has been renominated for a three year term. Born in Chicago, Ill., he graduated from the University of Chicago in 1920. Mr. Moulton has had extensive experience in the petroleum industry, served as assistant professor at University of South Dakota, and headed the petroleum section of the Illinios State Geological Survey. He was later employed by United Gas Co., Electric Bond & Share Co., and Ralph E. Davis, Inc. Late in 1951, he joined Rockefeller Brothers Inc., as a consultant. A former Vice Chairman of the Petroleum Branch, Mr. Moulton has served on many AIME committees and for the last year has been Treasurer.

RESEARCH IN PROGRESS SESSION

Institute of Metals Div.

Fall Meeting

A Research in Progress session is again being organized as a part of the technical program of the AIME Institute of Metals Div., for the forthcoming fall meeting in Philadelphia. This session is intended as in the past to provide an opportunity for the brief presentation of current research results which are considered by the investigator to be of sufficient interest to warrant public airing before the research is complete or ready for formal publication. All those interested in contributing to this program are invited to send in abstracts (about 200 words) of their proposed presentation to: Prof. Paul Gordon, dept. of metallurgical engineering, Illinois Institute of Technology, Technology Center, Chicago 16, Ill.

The deadline for the receipt of abstracts is set as Sept. 12, 1955. Presentation should be planned for a 5 min time limit plus 5 min for discussion. The abstracts will not be published but those selected for presentation will be available in mimeographed form at the registration desk during the October meeting.

Contributors are assured that selection will not be on an arbitrary basis nor on the basis of attempts to judge the relative usefulness or value of the papers submitted. Rather, papers will be chosen according to how well the contribution falls into the *Research in Progress* category and according to the possibility of grouping several reports around some unifying principle.

New Metals Section Holds Dinner Meeting

A dinner meeting of the Niagara Frontier Section was held at Niagara Falls, N. Y. on May 18, 1955. The speaker for the meeting was J. H. Hollomon, manager of the metallurgy and ceramics dept., General Electric Research Laboratory. Dr. Hollomon was recently honored as the outstanding young man in the country in the field of metallurgy.

In his talk entitled *Modern Metallurgy*, Dr. Hollomon first reviewed present industry to show that it is based upon the general principles of physical chemistry which were defined basically by laboratory studies made 50 years or more ago. At the present time, metallurgical laboratories are studying principles which will find many industrial applications in the coming years. The main subject of the talk was a review of present knowledge of the dislocations which occur during crystal growth. These dislocations are fundamental to processes of sintering, grain growth, transformation reactions, and the general properties of materials. Dr. Hollomon predicted that control of dislocations in metals, such as is already being done in transistors, will result in production of materials in the next 10 years with properties far exceeding those now in use. Looking ahead to the future of metallurgical research, Dr. Hollomon stated that the next step forward would involve putting the electron to work to determine the proper materials for use in a given application.

Interest resulting from Dr. Hollomon's excellent presentation was shown by the active discussion period led by R. W. Fountain, Technical Chairman of the meeting.

EJC Sponsors Nuclear Congress

Engineers Joint Council, sponsor of the Nuclear Engineering and Science Congress to be held in Cleveland, December 12 to 16, 1955, has received 375 suggested titles for papers to be presented at that meeting. A total of 292 titles have been approved, and it is expected that a large proportion will develop into papers that will be approved by the cooperating societies. Attendance is expected to be from 2000 to 3000, or twice that of the Ann Arbor Nuclear Congress held in 1954. Headquarters hotel will be the Statler, but substantially all papers will be presented in the meeting rooms of the Public Auditorium in Cleveland, where the Atomic Exposition will be held. The latter is being sponsored by the American Institute of Chemical Engineers and will be coordinated with the technical program. Registrants at the Congress will be admitted to the Exposition free at all times, but public admission will be restricted to Saturday and Sunday, the 10th and 11th, and to evenings. Copies of the papers will be sold at 30¢ each.

Request for Catalogs Received from Formosa

H. R. Hanley, lately of the Missouri Schools of Mines, and a member of the St. Louis Section, is in Formosa as advisor to the dept. of mining and metallurgy, Taiwan College of Engineering.

He is faced with the task of setting up a mining and metallurgical dept. from the school. In addition to setting up an instruction program, Professor Hanley, who is a Legion of Honor Member of the AIME, is also charged with procurement.

One of his greatest problems is the lack of catalogs. He is making a plea to manufacturers to fill the gap. Those who can help should respond directly to: Prof. H. R. Hanley, Purdue Team, c/o Army Section, Artillery School, APO 63, c/o Postmaster, San Francisco, Calif.

Professor Hanley seeks literature covering every conceivable item used in the metallurgical and mining fields.

— Coming Events —

Aug. 11-12, Denver Research Institute, Industrial Applications of X-Ray Analyses, annual symposium, University of Denver, Denver.

Aug. 22-23, Stanford Research Institute, Symposium on Electronics in Automatic Production, Stanford.

Sept. 6-16, Metal Show, Chicago Coliseum, Chicago.

Sept. 14-16, Porcelain Enamel Institute, annual shop practice forum, Deshler-Hilton Hotel and Ohio State University, Columbus.

Sept. 25-28, American Institute of Chemical Engineers, Lake Placid Club, Lake Placid, N. Y.

Sept. 26-30, Atomic Industrial Forum, trade fair, Sheraton-Park Hotel, Washington, D. C.

Oct. 9-13, Electrochemical Society, William Penn Hotel, Pittsburgh.

Oct. 17-19, AIME, IMD, fall meeting, Adelphia Hotel, Philadelphia.

Oct. 24-25, Steel Founders Society of America, fall meeting, Greenbrier, White Sulphur Springs.

Oct. 27-28, National Conference on Industrial Hydraulics, annual meeting, La Salle Hotel, Chicago.

Oct. 31-Nov. 2, National Lubricating Grease Institute, annual meeting, Edgewater Beach Hotel, Chicago.

Nov. 1-5, World Symposium on Applied Solar Energy, Westward-Ho, Phoenix.

Nov. 4, AIME, Pittsburgh Local Sections, off-the-record meeting, Pittsburgh.

Sept. 15, 1955 is the deadline for the Institute of Metals Div. papers for the 1956 New York Annual Meeting and the deadline for all Iron & Steel Div., and Extractive Metallurgy Div. papers to be preprinted. Papers received by this deadline but requiring revision may not be processed in time to permit scheduling for this meeting.

Personals

A. R. LYTLE

Arthur R. Lytle, Electro Metallurgical Co., has been appointed vice president in charge of research. Mr. Lytle has been associated with Union Carbide & Carbon Corp., since 1923.

Paul A. Lockwood, production metallurgist, Thompson Products, Cleveland, Ohio, has joined the Owens Corning Fiberglass Corp., Newark, Ohio, as a research metallurgist.

William D. Heavner, Jr., recently joined the research staff of Allegheny Ludlum Steel Corp., stainless steel research section, Pittsburgh, Pa.

Austin R. Zender, executive vice president, Bridgeport Brass Co., Bridgeport, Conn., was elected president of the Copper & Brass Research Assn., New York, N. Y.

Ernest C. Kron, formerly steel metallurgist, has been promoted to division metallurgist, Doehler-Jarvis div., National Lead Co.

Charles G. Beavers, Jr., was elected president of Follansbee Metals, Inc., Wallingford, Conn.

A. B. Miner was reelected president and chairman of the board, National Lime Assn. Mr. Miner is general sales manager of the industrial div., National Gypsum Co., Buffalo, N. Y.

Bernard Kopelman has been appointed chief engineer of the atomic energy div., Sylvania Electric Products Inc.

W. S. Lowe, president of A. P. Green Fire Brick, Co., Mexico, Mo., was elected president of the Refractories Institute, Pittsburgh, Pa.

George C. McKann, Jr., was recently appointed executive director of the National Metal Trades Assn., Chicago, Ill.

Byron E. Grant has been named assistant manager of the Braden Copper Co., div. of the Kennecott Copper Corp. Mr. Grant will be located in the Sewell-Rancagua area of Chile.

William L. Walsh has been appointed assistant to the president of Kennecott Copper Corp., New York.

C. A. Brashares has been named vice president, sales and **A. Brent Wilson** has been elected vice president, subsidiaries, Harbison-Walker Refractories Co., Pittsburgh. **W. C. Robinson, Jr.,** president of National Electric Products Corp., was elected a director of the company.

Roger M. Blough succeeded **Benjamin F. Fairless** as chairman of the board and chief executive of the U. S. Steel Corp. Mr. Fairless has retired, but will continue as a member of the board of directors and the finance committee of the corporation.

G. R. HEFFERNAN

G. R. Heffernan has been named managing director for the Premier Steel Mills Ltd., Edmonton, Alberta, Canada. Mr. Heffernan had been superintendent of the Vancouver Steel Co., Ltd., Granville Island, Vancouver, B. C.

William J. Harris, Battelle Memorial Institute, Columbus, Ohio, has been named assistant to the director and will be located in Washington, D. C. Dr. Harris has been associated with Battelle for approximately a year.

C. A. Stokes, director of research and development for Godfrey L. Cabot, Inc., Boston, Mass., has been elected to the board of directors of the Industrial Research Institute.

W. H. Sparr, International Nickel Co., Inc., has been transferred from Pittsburgh, Pa., to New York, N. Y.

Ivor D. Thomas is a metallurgist with General Electric Co., product metallurgy unit, engineering dept., at the Hanford atomic installation near Richland, Wash. Mr. Thomas was formerly plant manager, Induction Steel Foundry Co., Salt Lake City, Utah.

Norvall M. Hunthausen, district sales manager for the Mexico Refractories Co., Mexico, Mo., has been appointed assistant secretary and treasurer of the Southwest Section of the AIME National Open Hearth Committee. He succeeds **H. W. Gethin,** Laclede-Christy Co., who has been transferred to Pittsburgh.

L. W. Cope is a metallurgist at the Wah Chang Mining Corp., Lincoln mine, Tempiute, Nev. He had been associated with the U. S. Vanadium Co.

W. Maxwell Parris has joined the staff of the technical dept., metallurgical div., Titanium Metals Corp. of America, Henderson, Nev. Mr. Parris was formerly assistant chief, light metals div., Battelle Memorial Institute, Columbus, Ohio.

Donald J. McPherson has been promoted to assistant manager of the metals research dept., Armour Research Foundation of Illinois Institute of Technology, Chicago, Ill. Dr. McPherson was formerly supervisor of the dept.'s nonferrous metallurgy section.

Richard T. Fleming has retired as vice president and general counsel of the Texas Gulf Sulphur Co., New York.

Richard S. Moore, assistant general sales manager, Harbison-Walker Refractories Co., Pittsburgh, has been appointed general sales manager. He first became associated with the company in 1929.

R. S. MOORE

Robert L. Fischer, formerly with Boeing Aircraft Co., mfg. research dept., Seattle, Wash., has joined the Westinghouse Electric Corp., atomic power div., Pittsburgh, Pa., as a development engineer.

Harold L. Beynon has been promoted to assistant general sales manager by the A. P. Green Fire Brick Co., Mexico, Mo. Mr. Beynon had been regional sales manager.

Andrew Fletcher, president of St. Joseph Lead Co., New York, N. Y., was elected president of the Lead Industries Assn.

Ralph E. Carter was appointed a research associate in metallurgy by the General Electric Research Laboratory, Schenectady, N. Y.

Robert E. Wasser, assistant secretary, Wheeling Steel Corp., Wheeling, W. Va., has retired after 22 years in that capacity.

Cyrus J. Taylor, manager of National Gypsum Co.'s Bronx, N. Y. plant, has retired. **Richard G. Tarbell,** formerly plant manager of the Fort Dodge, Iowa plant has succeeded Mr. Taylor.

W. Roy Willard has been appointed assistant to the president, Buffalo Steel Corp., Tonawanda, N. Y.

T. K. Redden has joined the General Electric Co., A.G.T. development dept., materials laboratory, Cincinnati, Ohio. Mr. Redden had been with the Titanium Metals Corp. of America, New York, N. Y.

Carl W. Wagner, dept. of metallurgy, Massachusetts Institute of Technology, Cambridge, Mass., has been appointed to the rank of full professor. **Maurice E. Shank,** dept. of mechanical engineering, was promoted to the rank of associate professor.

Edward L. Thellmann, Horizons Inc., Cleveland, Ohio, has been promoted to project supervisor. He had been senior research associate.

Olof A. Sundstrom is metallurgical superintendent, Rhokana Corp. Ltd., Kitwe, Northern Rhodesia. Mr. Sundstrom had been general superintendent for Bolidens Gruvaktiebolag, Skelleftehamn, Sweden.

Robert Maddin, associate professor of metallurgy, Johns Hopkins University, Baltimore, Md., has been appointed professor of metallurgy in the school of metallurgical engineering, University of Pennsylvania, Philadelphia, Pa. Dr. Maddin will assume the professorship July 1.

Paul W. Marshall, U. S. Steel Corp., has been transferred to Monroeville, Pa., as assistant director in the applied research laboratory. He had been research associate in the research and development laboratory, Pittsburgh, Pa.

E. H. KINELSKI

Eugene H. Kinelski has joined the development and research div., International Nickel Co., Inc., as a research metallurgist in the welding section of the research laboratory at Bayonne, N. J.

Henry S. Schaufus recently became chief metallurgical engineer of Vanadium Corp. of America.

Charles Wicks Sherman, Latrobe Steel Co., Latrobe, Pa., has been promoted to director of research.

Harleston R. Wood has been elected president and chief executive officer of the Alan Wood Steel Co., Conshohocken, Pa. Mr. Wood will succeed **John T. Whiting,** who will continue as chairman of the board and a member of the board of directors.

Harold B. Emerick has been named director of technical services div., Jones & Laughlin Steel Corp., Pittsburgh, Pa. Mr. Emerick joined J&L in 1935 as a metallurgical inspector at the Aliquippa Works. In 1942 he was co-recipient of the McKune Award given by the AIME National Open Hearth Steel Committee.

Frank W. Glaser was appointed a vice president and director of the Alloy Precision Castings Co., Cleveland, Ohio. He was formerly vice president in charge of research and development, American Electro Metal Corp., Yonkers, N. Y.

W. E. Ferguson has joined the Lockheed Aircraft Corp., Burbank, Calif., as a process analyst. He had been with the U. S. Steel Corp., Columbia-Geneva div., Pittsburgh, Calif.

H. R. Youngkrantz, chief engineer, Apex Smelting Co., Chicago, Ill., has been transferred to Cleveland, Ohio.

Morris Boorky, president, Presmet Corp., Worcester, Mass., was elected president of the Metal Powder Assn. **B. T. duPont,** sales manager, plastic metals div., National-U. S. Radiator Corp., Johnstown, Pa., was elected chairman of the board of the association.

Robert A. Eidam is now a special apprentice for the Continental Foundry & Machine Co., East Chicago, Ind. He had been associated with American Steel Foundries, Hammond, Ind.

W. L. Bruckart, a former member of the staff at Battelle Memorial Institute, Columbus, Ohio, has accepted a position with the Universal-Cyclops Steel Corp., Bridgeville, Pa.

Arthur B. Michael recently joined the research dept. of the Fansteel Metallurgical Corp., North Chicago, Ill. Dr. Michael had been a research engineer with Allis-Chalmers Mfg. Co., Milwaukee, Wis.

David L. Wheeler is a trainee with the Kaiser Aluminum & Chemical Corp., Newark, Ohio. He recently transferred from ferrous to aluminum metallurgy.

Charles N. Treat, Republic Steel Corp., had been general foreman at Chicago, Ill. Mr. Treat is now superintendent at Moline, Ill.

James McIntosh has joined the Iron King mine, Shattuck-Denn Mining Corp., Humboldt, Ariz., as a metallurgist.

Robert Steinitz, physicist at American Electro Metal Corp., Yonkers, N. Y., has been appointed director of research.

Earl W. Ganslein is now associated with the Buckeye Steel Castings Co., Columbus, Ohio. He had been with the American Steel Foundries, Alliance, Ohio.

Kent R. Van Horn, director of research for the Aluminum Co. of America, received an honorary degree of Doctor of Science from Case Institute of Technology. Dr. Van Horn is a graduate of the 1926 class of Case.

S. W. Boone was appointed manager of public relations of the tubular products div., Babcock & Wilcox Co., Beaver Falls, Pa. Prior to this appointment, Mr. Boone was staff assistant to the works manager at the Beaver Falls plant.

Henry Mahlstedt has been named manager of sales for the electroplating & castings sales dept., united chromium div., Metal & Thermit Corp. **Donald R. Meserve** becomes assistant manager of sales.

Henry S. Wingate, president of the International Nickel Co. of Canada Ltd., was appointed a director of the Bank of Montreal. Mr. Wingate joined Inco 20 years ago and became president of the company and its U. S. subsidiary in 1954.

John W. Swengel has been appointed works manager of the R-P&C valve div., American Chain & Cable Co., Reading, Pa. He formerly served as foundry superintendent.

S. A. Ambler, assistant works industrial engineer, Jones & Laughlin Steel Corp., continuous galvanizing dept., Pittsburgh Works div., has been promoted to general foreman. **J. C. Hawkins,** former general metallurgical investigator has been named assistant general foreman. **D. W. Ferguson,** assistant general foreman, electrical, has been promoted to master mechanic.

Francois J. Olmer, Armour Research Foundation of Illinois Institute of Technology, has been promoted to senior chemist. Dr. Olmer joined the Foundation as a research physical chemist in 1952. He had been in charge of special development work for the Celanese Corp. of America, Summit, N. J.

C. G. Hogberg has been appointed assistant to vice president of the Michigan limestone div., U. S. Steel Corp., and he will be located in Detroit. Mr. Hogberg has been associated with U. S. Steel since 1935. Mr. Hogberg is a member of the Executive Board of the AIME Blast Furnace, Coke Oven and Raw Materials Committee. In 1945 he was recipient of the J. E. Johnson Award.

Donald E. Wingate was named chief engineer for Ipsen Industries, Inc., Rockford, Ill. He had been associated with Surface Combustion Corp.

William V. Burley, Jr., has been appointed comptroller of Titanium Metals Corp. of America, New York.

Arnold H. Miller, consulting engineer of New York, is at present doing examination work in Mexico.

Robert F. Ames has been named New York purchasing representative, U. S. Steel Corp. Mr. Ames joined U. S. Steel in 1934 at South Chicago Works.

Andrew E. St. John has been appointed technical manager of Barth Smelting Corp., Newark, N. J.

L. M. Petryck has joined the development and research div., International Nickel Co., Inc., as a research metallurgist in the welding section of the research laboratory at Bayonne, N. J.

William A. Griffith has been transferred from the research dept., New Jersey Zinc Co., Palmerton, Pa., to the Bertha mineral div., Austinville, Pa.

Whitman G. Rouillard is manager of American Smelting & Refining Co.'s Tacoma, Wash., smelter. He was formerly manager of the Garfield plant, Salt Lake City, Utah.

C. L. Kingsbury, Jr., Reserve Mining Co., Babbitt, Minn., is superintendent of crushing and concentrating operations at the Babbitt plant. Mr. Kingsbury joined Reserve in 1951 as a field engineer.

William A. Seedorff has been promoted to project engineer, International Minerals & Chemical Corp., Carlsbad, N. M.

Harold E. Rowen is general manager of the new Dwight-Lloyd div. of McDowell Co., Inc., Cleveland. He was vice president of Sintering Machinery Corp., Netcong, N. J., recently acquired by McDowell Co., Inc.

Donald E. Moat has been appointed assistant director of marketing of the Leeds & Northrup Co. Mr. Moat was formerly district sales manager at the Cleveland offices.

Charles W. Mote was named plant manager of the Dodge div., forge plant, Chrysler Corp., Detroit. Mr. Mote joined the Dodge forge plant in 1933.

Benjamin F. Shepherd, chief metallurgist, Ingersoll-Rand Co., Phillipsburg, N. J., recently received an honorary degree of Doctor of Science from Lafayette College.

William N. Brinker has been promoted to market development manager, Porcelain Enamel Institute, Washington, D. C.

Bert L. Wood, consulting engineer, American Iron & Steel Institute, New York will retire on October 1. Mr. Wood has served the steel industry for more than 40 years. He will continue in an advisory capacity as a part-time consultant. **William G. Kirkland** will succeed Mr. Wood. Mr. Kirkland joined the staff of AISI in 1947.

John L. Reuss is a metallurgical engineering trainee with Oliver iron mining div., eastern district, Virginia, Minn.

John L. Splane, general superintendent, U. S. Graphite Co., and Cia. Minera de San Jose, Moreno, Sonora, is now with San Francisco Mines of Mexico Ltd., San Francisco del Oro, Chihuahua.

J. L. SPLANE

C. Kremer Bain has retired from St. Joseph Lead Co., Bonne Terre, Mo., to enter private consulting work.

Lois G. McDowell, assistant professor of mechanical engineering, Illinois Institute of Technology, Chicago, has been elected national president of the Society of Women Engineers for 1955-56.

George B. Beitzel has retired as president of the Pennsylvania Salt Mfg. Co., but will continue to serve as a member of the board and will become chairman of the board of the company's foreign operations subsidiary.

L. W. Buechler has been named chief engineer for Vickers electric div., Vickers Inc., St. Louis, Mo. Mr. Buechler was formerly manager of control product sales for the div.

Roger H. Watkins has joined the Cleveland sales staff of F. J. Stokes Machine Co., Philadelphia, Pa., as senior sales engineer.

Emo D. Porro has been appointed engineering assistant to the director, Stanford Research Institute, Stanford, Calif. He has been a staff member since 1948. Previous to that he had been assistant to the vice president, Kaiser Services, Oakland, Calif.

Hamilton Gray, professor of civil engineering, University of Maine, has been appointed chairman of the dept. of civil engineering, college of engineering, Ohio State University. Professor Gray will assume has duties on October 1.

Thomas E. Williams has been named general superintendent of West Coast plants, Joseph T. Ryerson & Son, Inc. This will be in addition to his duties as superintendent of the Los Angeles steel service plant.

Richard L. Templin, assistant director of research, Aluminum Co. of America, has been awarded an honorary membership in the American Society for Testing Materials. This is the highest award the society bestows.

Don Palmisano has been promoted from service representative to southern sales representative, electrode div., Great Lakes Carbon Corp., New York. Mr. Palmisano joined Great Lakes in 1951.

Harold Nutt was elected president and general manager of the Borg & Beck div., Borg-Warner Corp., Chicago, Ill. He was previously vice president and general manager.

I. D. Thomas formerly associated with Induction Steel Foundry Co., Salt Lake City, has joined the General Electric Co., Richland, Wash., as a research metallurgist.

Peter Zouraeff, Reynolds Alloys Co., Listerhill, Ala., is now with Reynolds Metals Co., Richmond, Va.

Obituaries

J. W. R. Husen (Member 1953) died suddenly of polio in Marrakech, French Morocco, Mar. 23, 1955. Mr. Husen, representative of the Billiton Co. in Morocco, was born in Holland in 1916. After receiving his technical training from the Technische Hoogeschool in Delft, he was assistant to Prof. H. F. Grondijs in his economic geology and ore dressing laboratory. Mr. Husen was later chief engineer and then director of mines, Societe Internationale d'Exploitation Miniere au Mauroc in Casablanca. He was also consulting mining engineer, Societe Epermines in Casablanca, and more recently director, Societe O.G.I.M. in Oued-Zim, French Morocco.

John W. H. Hamilton (Member 1912) died in Sweden on Jan. 11, 1955. Mr. Hamilton was born in 1878 at Kristianstad, Sweden and was a graduate of the Royal Institute of Technology, Stockholm. He was engaged in post graduate work in Germany. Mr. Hamilton came to the U. S., and from 1900 to 1904 was employed by the General Electric Co., Schenectady, N. Y., in the testing dept., factory and drafting dept. He returned to Sweden for a short time and then became a partner in Gillberg & Hamilton Engineers, New York, N. Y. In 1906 he became a partner in Hamilton & Hansell.

Robert D. Pike (Member 1942) died on April 13. A native of San Francisco, Calif., Mr. Pike attended the Universities of Stanford and California. He had been a consulting chemical engineer and worked on various projects. He developed and manufactured a special high grade synthetic substitute for Austrian magnesite during the first world war; studied utilization of resources of Wyoming, Idaho, Utah, and Montana; worked on development of gravelite lightweight concrete; and also did research into the manufacture of magnesite from dolomite.

I. J. Simcox (Member 1932) died of a heart attack on Feb. 13, 1955. He was born at Hubbard, Ohio in 1891. He was a graduate of Pritchett College and Missouri School of Mines and Metallurgy. He was first employed by the Anaconda Copper Mining Co. at Anaconda, Mont. After a short association with the Research Corp., New York, N. Y., Mr. Simcox joined the Phelps Dodge Corp., Douglas, Ariz. He remained with this firm for 9½ years and then joined the metallurgical staff of the International Nickel Co., Copper Cliff, Ont., Canada. Two years of his 23½ with Inco were spent in the Petsomo district in Finland. At the time of his retirement, Mr. Simcox was assistant to the general manager.

Necrology

Date Elected	Name	Date of Death
1952	G. E. Allen	Unknown
1916	Barclay G. Anderson	May 16, 1955
1940	Crosby F. Barker, Jr.	Unknown
1948	John G. Bartram	Feb. 1955
1918	George B. Bassett	Unknown
1912	Quincy Bent	May 5, 1955
1901	James F. Brophy	Unknown
1914	W. C. Campbell	Unknown
1930	Ermil Leslie Caster	Apr. 22, 1955
1948	A. Clemes	Dec. 4, 1954
1950	William H. Cooke	Mar. 29, 1954
1935	C. A. Heberlein	May 6, 1955
1908	Arthur E. Hepburn	Unknown
1943	Lafayette Herring	Mar. 3, 1955
1920	Dudley Denison Homer	May 16, 1955
1937	Julian Kennedy, Jr.	May 5, 1955
1913	Karl L. Landgrebe	May 13, 1955
1953	Paul G. Leroy	Feb. 13, 1955
1902	John H. Lidgerwood	Mar. 9, 1955
1903	J. F. McClelland	May 5, 1955
1941	John H. McElhinney	Apr. 15, 1952
1949	Jean Model	May 29, 1955
1954	Harry B. Rose	Unknown
1901	H. P. Townsend	Feb. 17, 1955

Proposed for Membership
— Metals Branch AIME —

Total AIME membership on May 31, 1955 was 22,664; in addition 1804 Student Associates were enrolled.

ADMISSIONS COMMITTEE

P. D. Wilson, Chairman; F. A. Ayer, Vice-Chairman; A. C. Brinker, R. H. Dickson, T. D. Jones, F. T. Hanson, Sidney Rolle, O. B. J. Fraser, F. T. Sisco, Frank T. Weems, R. L. Ziegfeld, R. B. Caples, F. W. McQuiston, Jr., A. R. Lytle, H. R. Wheeler, L. P. Warriner, J. H. Senff.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

California
Anaheim—Bernstein, Leonard (J)
Playa Del Rey—Bittmann, Charles A. (M) (C/S—A-M)

Illinois
Chicago—Miller, Robert W. (A)

Indiana
Indianapolis—Wolff, Ursula E. (M)
Lafayette—Guy, Albert G. (M) (R. C/S—J-M)

Massachusetts
Boston—Parlanti, Conrad A. (M)
Cambridge—Mesick, Benjamin S. (M)

Michigan
Royal Oak—Stefanski, Henry F. (M)

Montana
Great Falls—Koza, Stephen (A)

New Jersey
Butler—Brown, Clayton D. (M) (C/S—A-M)
Rumson—Lawrence, Kenneth L. (M)

New York
Harrison—Abrams, Arthur J. (J)
Lewiston—Hurd, Frank W. (M)
Lewiston—Wyche, Ernest H. (M)
Snyder—Andersen, Harold A. (R. C/S—S-A)
Snyder—Guarnieri, Glen J. (M) (R. C/S—A-M)
Westport—Pratt, Charles H. (M)

Ohio
Parma—Sobey, Edmond J. (M) (C/S—A-M)

Pennsylvania
Elkins Park—Thalheimer, John M. (J)
Latrobe—Hamaker, John C., Jr. (M) (C/S—S-M)
New Kensington—Russell, Allen S. (M)
Pittsburgh—Collins, John F. (M) (C/S—J-M)
Pittsburgh—Cornelius, Nelson C. (M)
Pittsburgh—Forscher, Frederick (M)
Sharpsville—Swym, William B. (M)

Washington
Fruitland—Lower, James W. (M)

West Virginia
Wheeling—Hansen, George P. (M) (R. M)

Canada
Ont., Hamilton—Sibakin, Jaroslaw G. (M)

Mexico
Mexico City—Arze, Jose S. (J)

Peru
La Oroya—Hull, Kenith (M)

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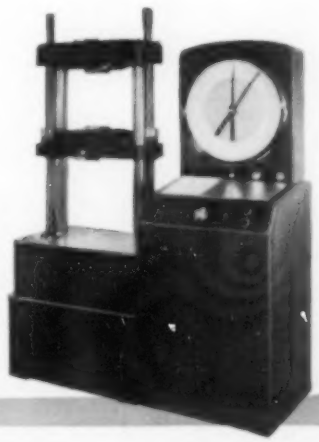

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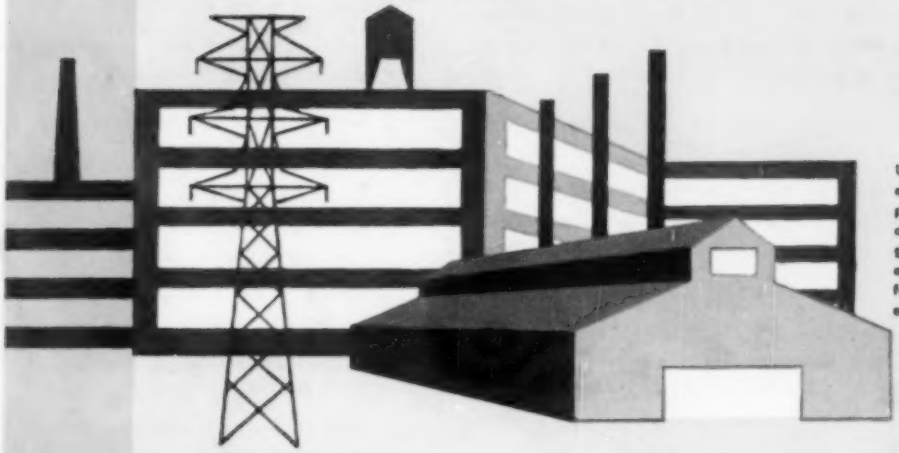
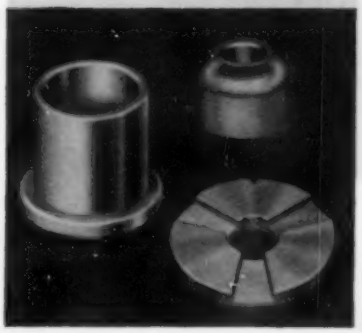


USG® Brushes are available in production or replacement lots in carbon, carbon-graphite, electrographitic, pure graphite and metalgraphite grades for all types of rotating electrical equipment.

GRAPHITAR® (carbon-graphite) is a versatile engineering material of carbon-graphite composition offered in many grades for bearings, seals, piston rings, pump vanes, thrust washers, valve seats, pistons, bushings.



GRAMIX®: Sintered-metal bearings, gears, cams, slides, rollers, bushings and machine parts available in ferrous and non-ferrous alloys. The metal with the built-in lubrication.

Metallurgical graphite grades for carbon control in iron and steel melting, mold coatings, hot topping compounds, Cerdip®, Mexaloy® for refractory mixtures and other MEXICAN® products.

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